MINING METHODS AND COSTS AT THE MORNING MINE, AMERICAN SMELTING & REFINING CO., SHOSHONE COUNTY, IDAHO

BY JOHN R. REYNOLDS

Information Circular 7743

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John R. Reynolds[1/]

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1/ Formerly mining engineer, Bureau of Mines.

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INTRODUCTION AND SUMMARY

This paper describes mining and milling methods of the American Smelting & Refining Co. at the Morning mine in the Mullan area of the Coeur d'Alene mining region in Idaho and is one of a series of Bureau of Mines reports dealing with methods and costs of mining throughout the United States. The Morning mine has been described previously in a Bureau of Mines report published in 1930,[2/] and the milling methods and costs at the Morning concentrator were presented in a Bureau of Mines report published in 1932.[3/] These reports are supplemented by this publication, which presents data to show the effect of changing times and conditions on a typical medium-size mining operation during the latter part of its history, when costs were rising and production was from increasingly greater depths.

The report contrasts two periods in the history of the mine—periods not too far separated in years but considerably different in economic conditions. During the first period (1939) costs of labor and materials were moderate, mining operations were just being transferred from the main shaft to deeper workings opened from the offset shaft, and the operation prospered. During the second period (12 years later) production costs had increased following the national trend in all industry, mining was from greater depths near the bottom of the ore body, and the operation was becoming unprofitable. Mining on the lower levels of the property ceased in 1953.

The Morning mine has been one of the largest lead-silver-zinc mines in the United States, and the ore shoot is reputed to have the world's greatest dip length. The lowest level (5,200-foot) is approximately 4,200 feet below creek level and 6,700 feet below the outcrop.

ACKNOWLEDGMENTS

Grateful acknowledgment is made to J. E. Berg, general manager, Northwestern Mining Department, American Smelting & Refining Co. for permission to obtain and publish this information and to his assistant, J. C. Kieffer, for his help in preparing it.

2/ Wethered, C. E., and Coady, L. J., Mining Methods at the Morning Mine of the Federal Mining & Smelting Co., Mullan, Idaho: Bureau of Mines Inf. Circ. 6238, 1930, 13 pp.

3/ Dalton, M. P., Milling Methods and Costs at the Morning Concentrator of the Federal Mining & Smelting Co., Mullan, Idaho: Bureau of Mines Inf. Circ. 6587, 1932, 11 pp.

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Particularly valuable were the assistance and cooperation of W. J. Coombe, superintendent of mines, H. H. Shook, mine superintendent, Morning mine, G. S. Price, superintendent of concentrators, A. E. Johnson, resident engineer, Morning mine, P. I. Conley, geologist, and W. J. Murphy, accountant.

Information published by the Federal Geological Survey[4/] and the Federal Bureau of Mines[5/] was freely drawn upon in preparing this paper.

HISTORY AND PRODUCTION

The Morning lode, which outcrops about 2 miles north of Mullan, Idaho, was located July 3, 1884, by George S. Good. Early in 1886 Good deeded a quarter interest in the claim to John A. Carlson in lieu of payment for driving a 65-foot adit and later in the same year he sold an additional quarter interest to Carlson for $100. Toward the end of 1886, Good and Carlson sold the claim to Lewis Martin, F. M. Frank, and Charles Hussey for $1,200. Hussey, who bought out Martin and Frank in October 1887, patented the Morning claim in 1891.

The property, which by then comprised several claims in addition to the Morning and included the You Like mine, was purchased in 1892 by the Morning Mining Co., a Wisconsin corporation, for $400,000.

Peter Larson and Thomas Greenough purchased the property from the Morning Mining Co. in 1897 for $320,000 and developed it into one of the important producers of the district. A narrow-gage railroad was constructed up Mill Creek to the portal of the No. 5 tunnel, and a 600-ton mill was built. The No. 6 tunnel was begun in 1900 and was finished in 1906.

In October 1905 the property was purchased from Larson and Greenough for $3,000,000 by the Federal Mining & Smelting Co.; in 1953 Federal was absorbed by its parent company, the American Smelting & Refining Co.

On October 3, 1953, the following notice was posted on bulletin boards at the Morning mine:

> DUE TO INCREASED COSTS OF LABOR AND SUPPLIES,
> DIMINISHING ORE RESERVES, AND LOW METAL PRICES
> THE MORNING MINE WILL BE CLOSED PERMANENTLY
> AT 7 O'CLOCK A. M., SUNDAY, OCTOBER 4, 1953
>
> AMERICAN SMELTING & REFINING CO.

This notice applied only below the 1,050-foot level, the producing portion of the mine. Following the posting of the notice, the mine was stripped of salvageable equipment below this level.

4/ Ransome, F. L., and Calkins, F. C., The Geology and Ore Deposits of the Coeur d'Alene District, Idaho: Geol. Survey Prof. Paper 62, 1908, 203 pp.

5/ Work cited in footnote 2.

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Current operations are confined to extracting several large shaft pillars above the 1,050-foot and No. 6 tunnel levels, which the management estimated would be completed some time in 1955. No pumping is being done from the Morning mine. On the Star equivalent of the Morning 4650-foot level, a drill-hole connection has been made between the two mines and water from the Morning is pumped through the Star mine. Flow from the Morning to the Star is controlled by a valve on a pipe grouted in the drill hole.

The Morning mine has been in almost continuous production since 1887 with only occasional shutdowns during periods of labor unrest, and once in 1932 the mine was closed for 6 months because of low metal prices. From 1906 to 1940 the mine produced about 1,000 tons of ore per day, but since 1940 the production has gradually fallen to the present rate of less than 200 tons per day.

Bureau of Mines records give the total production of the mine from 1895 through 1952, including ore mined by the company, ore produced by lessees, dumps milled, and old tailings milled, as 14,136,333 tons, having a metal content of 1,008,940 tons of lead, 399,900 tons of zinc, and 40,857,780 ounces of silver. Data on production before 1895 are not available.

Table 1, compiled from the records of the Federal Mining & Smelting Co., gives the annual production of company-mined ore from 1915 through 1952 and average metal prices for each year. It shows the changing tenor of the ore, decreasing lead and silver values, and increasing zinc values as mining was carried deeper.

LOCATION AND PHYSICAL FEATURES

The Morning group of 45 patented and 6 unpatented claims is in the Hunter mining district of the Coeur d'Alene mining region, Shoshone County, Idaho. The Morning vein outcrops on the steep ridge separating Mill Creek from Grouse Creek about 2 miles north of the town of Mullan (fig. 1). The mill, the surface plant, and the portal of the 10,000-foot-long No. 6 tunnel, main haulage tunnel, are along the South Fork of the Coeur d'Alene River just west of the city limits, as shown in figure 2.

Mullan, a town of 2,300 population on U. S. Highway 10, is served by a branch line of the Northern Pacific Railway Co.

The mine is in an area of high relief, with ridges that rise steeply 2,000 to 3,000 feet above the valleys and streams with steep gradients. Annual precipitation is about 42 inches, a large amount of which falls during winter as snow.

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TABLE 1. - Annual production of company-mined ore and average metal prices, 1915-52

Year	Ore extracted, tons	Average content of ore mined			Average prices of metals		
		Lead, percent	Silver, oz.	Zinc, percent	Lead, per pound	Silver, per oz.	Zinc, per pound
1915	258,340	10.3	4.3	5.5	$0.04126	$0.5033	$0.09750
1916	476,156	7.9	3.9	5.2	.05065	.6775	.09570
1917	469,079	8.2	4.0	4.6	.06138	.8168	.08850
1918	265,877	8.2	4.2	4.4	.07002	.9944	.07670
1919	172,337	7.5	3.7	5.1	.05717	1.1077	.06650
1920	314,027	7.2	3.4	3.6	.07947	1.0980	.07442
1921	355,773	8.4	3.9	4.2	.04530	.9938	.04750
1922	305,960	9.1	3.5	4.9	.05719	.9952	.05716
1923	295,728	10.5	4.4	5.1	.07248	.7876	.06607
1924	342,924	9.4	4.4	4.7	.08083	.6678	.06344
1925	399,679	9.5	4.3	5.3	.09016	.6906	.07622
1926	423,881	9.4	4.1	5.5	.08417	.6211	.07337
1927	398,523	9.4	4.8	5.7	.06755	.5637	.06242
1928	391,669	9.2	4.3	5.8	.06306	.5817	.06027
1929	380,660	9.3	3.7	6.4	.06829	.5262	.06507
1930	361,372	9.0	3.4	6.5	.05503	.3750	.04428
1931	231,619	9.1	3.5	6.5	.04282	.2849	.03657
1932	102,352	9.8	3.8	6.7	.03214	.2784	.02746
1933	197,257	9.4	3.6	7.1	.03902	.3599	.04152
1934	220,022	8.8	3.3	7.1	.03865	.6425	.04178
1935	298,167	9.1	3.4	6.5	.04060	.7237	.04403
1936	313,866	8.2	3.1	6.5	.04667	.7700	.04919
1937	350,609	7.9	2.9	6.3	.06074	.7700	.06472
1938	262,329	8.1	2.9	6.0	.04791	.64314	.04500
1939	363,122	7.6	2.9	5.7	.05088	.67376	.05138
1940	336,603	7.7	3.4	6.5	.05179	.70625	.06332
1941	270,787	7.3	3.1	7.2	.05783	.70625	.07520
1942	236,902	7.1	2.7	7.9	.06471	.70625	.08250
1943	178,122	6.5	2.3	8.2	.06500	.70625	.08250
1944	112,838	6.2	2.2	7.3	.06500	.70625	.08250
1945	76,447	6.7	2.3	8.6	.06500	.70657	.08250
1946	56,945	6.4	2.0	6.7	.07811	.79697	.08889
1947	70,946	6.0	1.9	6.4	.14734	.90000	.10500
1948	71,261	5.9	1.8	6.3	.18413	.90000	.13589
1949	87,757	6.9	2.1	8.2	.15364	.90000	.12144
1950	107,890	6.5	2.0	8.5	.13296	.90000	.13866
1951	88,817	5.7	1.8	8.3	.17500	.90000	.18000
1952	82,908	5.9	2.0	8.1	.16467	.90000	.16215

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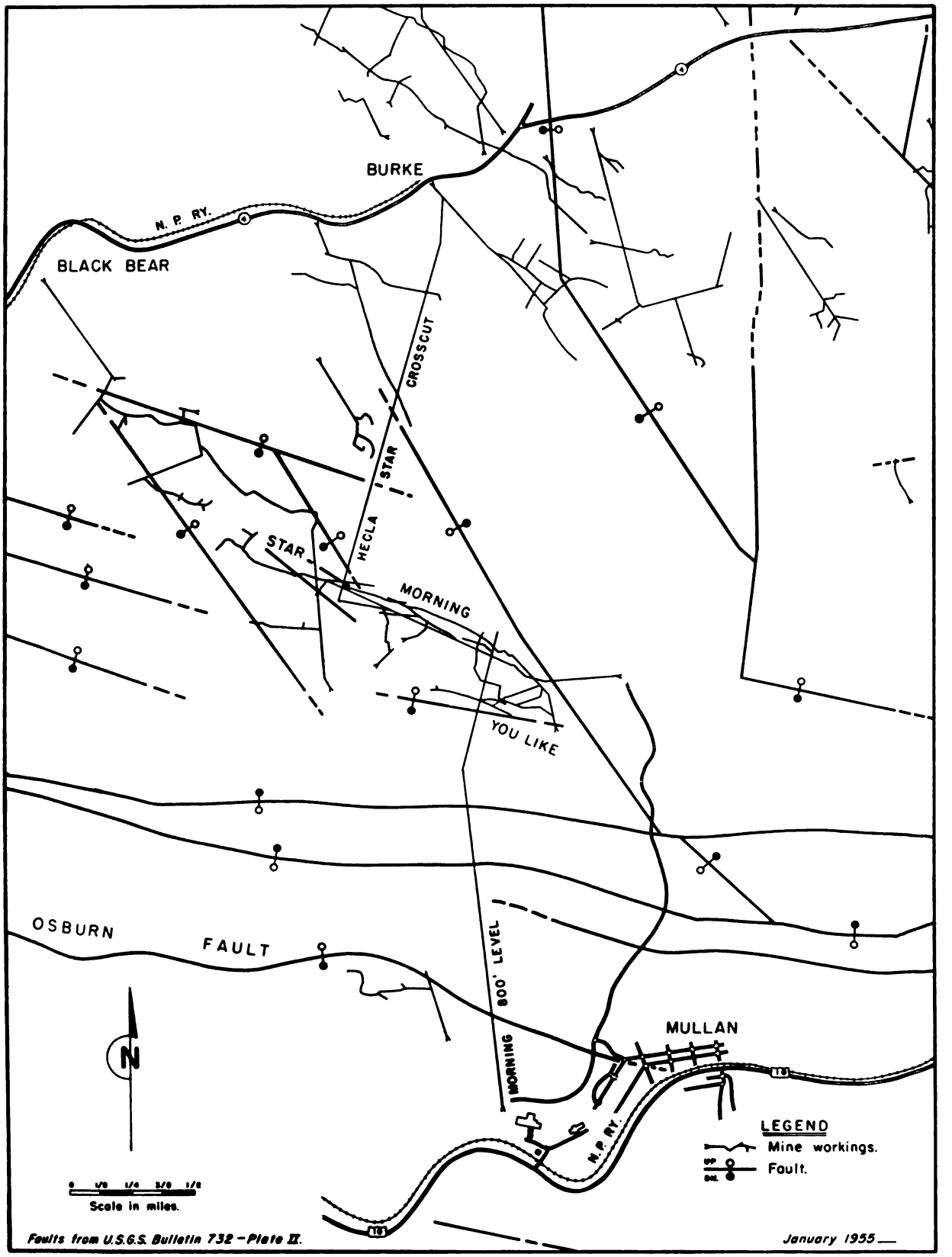

Figure 1. - Location of the Morning mine.

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Figure 2. - Mill and surface plant of the Morning mine, Shoshone County, Idaho.

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GEOLOGY AND PHYSICAL CHARACTERISTICS OF THE ORE

Rocks of the Coeur d'Alene district are mainly pre-Cambrian quartzites and argillites of the Belt series, which has been subdivided into the six formations listed below, in order of deposition.

Striped Peak formation	Interbedded flaggy quartzite and argillite
Wallace formation	Interbedded limy argillite and quartzite
St. Regis formation	Purple-gray and green argillite and argillaceous quartzite
Revett formation	Pure white quartzite
Burke formation	Light-green, impure quartzite
Pritchard formation	Black argillite

Although the base of the section is not exposed, the total thickness of these formations is estimated at more than 20,000 feet. These rocks have been intruded by granitic stocks and acidic and basic dikes, all of subordinate areal extent.

The dominant structural feature, the Osburn fault, traverses the district in a southeasterly direction and roughly divides it into north and south halves. Displacement along the fault has been estimated at 12 miles horizontally and 2,000 to 3,000 feet vertically, the north side having moved east and up relative to the south side.

The Morning Star ore body in the eastern part of the district lies on the complexly faulted and folded west limb of the broad south-plunging Granite Peak synclinorium between 7,000 and 8,000 feet north of the Osburn fault. The You Like vein system, of secondary importance, is parallel to and roughly 1,000 feet south of the Morning vein.

At its outcrop the Morning vein cuts the thin, argillaceous beds of the St. Regis formation and the quartzitic beds of the Revett formation, both of which strike generally northward and dip steeply eastward. The strike of the Morning vein varies from N. 65° W. to N. 80° W., and its dip varies from 85° N. to 80° S.

The transitional contact between the St. Regis and Revett formations is near the boundary between the Morning and Star mines, the Morning end of the ore body lying in the St. Regis formation, from which the bulk of the Morning mine production has come. Below the 3,650-foot level in the Morning and everywhere in the Star mine the ore body lies entirely within the Revett formation. The major part of the Morning Star ore body as a whole, therefore, was in the Revett.

Near the surface the Morning vein is a zone of close sheeting cutting the highly contorted beds at generally large angles. Mineralization follows the more intense fissuring. In the lower workings the sheeting was not so conspicuous, the stresses apparently having been confined to individual fractures. This difference

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in the character of the vein probably was due mainly to the difference in wall rock, inasmuch as the Revett formation is much more competent than the St. Regis formation.

The Morning ore shoot was continuous from its outcrop to and below the lowest level in the mine (5,200-foot), a vertical distance of more than 6,700 feet (fig. 3). This ore body was mined over a maximum strike length of about 2,000 feet in Morning ground and is currently being mined in the adjoining Star mine of the Sullivan Mining Co. On the Morning 1,650-foot level and the corresponding 3,300-foot level of the Star, the ore shoot has been stoped continuously over a strike length of 3,500 feet. Ore widths in the Morning varied from 3 to 40 feet, with an average stoping width of about 13 feet. The southeast limit of the ore body raked steeply northwest and as projected will cross the Morning Star end line approximately 1,000 feet below the 5,200-foot level.

In the upper workings and on the Star side of the end line, the Morning vein often was broken into several branches. This tendency to split or branch was apparently not so well developed in the lower levels, although several minor footwall splits yielded some ore.

The ground was highly fractured, and the ore broke easily; large fragments were seldom encountered in mining. The ore was not heavy so the timbers bore very little vertical weight; but there was tremendous side pressure, probably the result of unrelieved stresses in the sheared rock.

Galena and sphalerite, the principal ore minerals, occurred in small veins along fractures and foliation and in irregular bunches at intersections of fissures. Galena was dominant in the upper workings, and sphalerite was dominant in the lower Morning workings and on the Star end of the ore body. A typical vein section contained at least one band of solid high-grade ore from 1 to several feet thick having numerous stringers on both sides. The gangue was altered wall rock, quartz, and siderite,the latter two occurring much more abundantly in the upper levels. The relict vein rocks and the walls were highly sericitized and locally sideritized, this alteration having taken place more along selvage between folia and in gouge along faults and cleavage planes.

The You Like, a vein system of secondary importance,was a comparatively tight structure along which very little gouge had been developed. The ore minerals in order of economic importance were tetrahedrite, galena, and sphalerite.

SAMPLING AND ESTIMATION OF ORE RESERVES

At the Morning mine an experienced observer could tell by visual inspection whether or not a face contained ore of commercial grade. All drifts on the vein were sampled at 10-foot intervals and the samples assayed for lead, silver, and zinc. Ore-reserve calculations were made, using the assays of these samples and the vein widths in the drifts. The grade of the ore changed very little from level to level, and stopes were sampled infrequently; the few samples taken were not used in calculating ore reserves.

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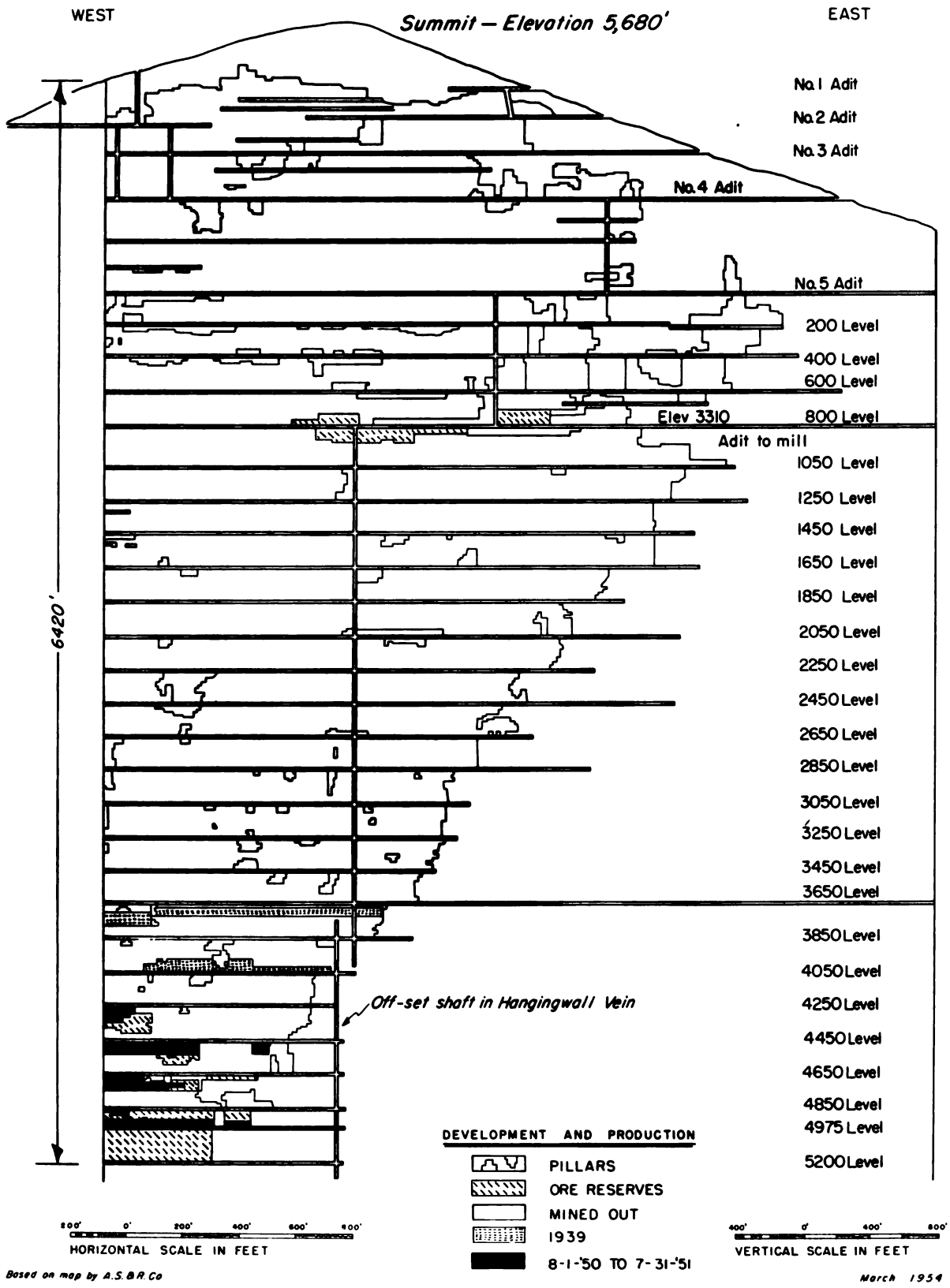

Figure 3. - Longitudinal projection of the Morning mine workings.

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The management estimates that 80,000 tons of unmined ore remained between the 5,200- and 5,000-foot levels when the lower levels of the mine were shut down.

METHOD OF DEVELOPMENT, MINING, AND MILLING

The Morning ore body was developed by 5 adits and a number of intermediate levels above the No. 6 tunnel, the main haulage level, and below by 22 levels. The level numbering sequence is shown in figure 3.

The collar of the main Morning shaft is about 110 feet in the footwall (south wall) of the vein and about 10,000 feet north of the portal of the No. 6 tunnel. This shaft is vertical and 4-compartment with levels 200 feet apart from the 1,050- to the 3,850-foot level. Main shaft stations and skip pockets were cut on opposite sides of the shaft, and from each a crosscut was driven to the vein. Figure 4 illustrates a typical main shaft station.

A vertical, 3-compartment offset shaft about 150 feet in the hanging wall (north wall) was begun in 1937 from the 3,850-foot level and was eventually extended to the sump below the 5,200-foot level. Levels were established every 200 feet except for the 150-foot interval between the 4,850- and 5,000-foot levels. The long axis of the offset shaft is roughly north-south with stations on the east side of the shaft and only one crosscut to the vein on each level. A generalized plan of an offset shaft station is shown in figure 5.

After the stations were cut and the crosscuts to the vein completed, drifts were driven on the vein to the limit of the ore on the east and to the property line on the west. These drifts were carried the full width of the vein and required heavy timbering.

As soon as the eastern end of the ore shoot or the west end line was reached, a ventilation raise was driven in either the footwall or hanging wall of the vein to the level above.

Because of prevailing ground conditions and consequent expense of maintaining all openings, stoping was begun immediately upon completion of the drift. Development was never far ahead of mining. Total mine openings in the Morning mine exceed 40 miles.

Immediate filling of stopes was necessary. Most of the waste for filling came from lateral drifts, ventilation raises, waste crosscuts, and raises in the walls.

Although the mine did not make an excessive amount of water, pumping was an item of considerable expense because of the necessity of maintaining 3 major and 6 small pumping installations, as well as an extensive conduit system. Water coming into the mine averaged 350 gallons per minute during the 3 months of the wet season, about 150 gallons per minute during the 3 dry months, and about 250 gallons per minute during the rest of the year.

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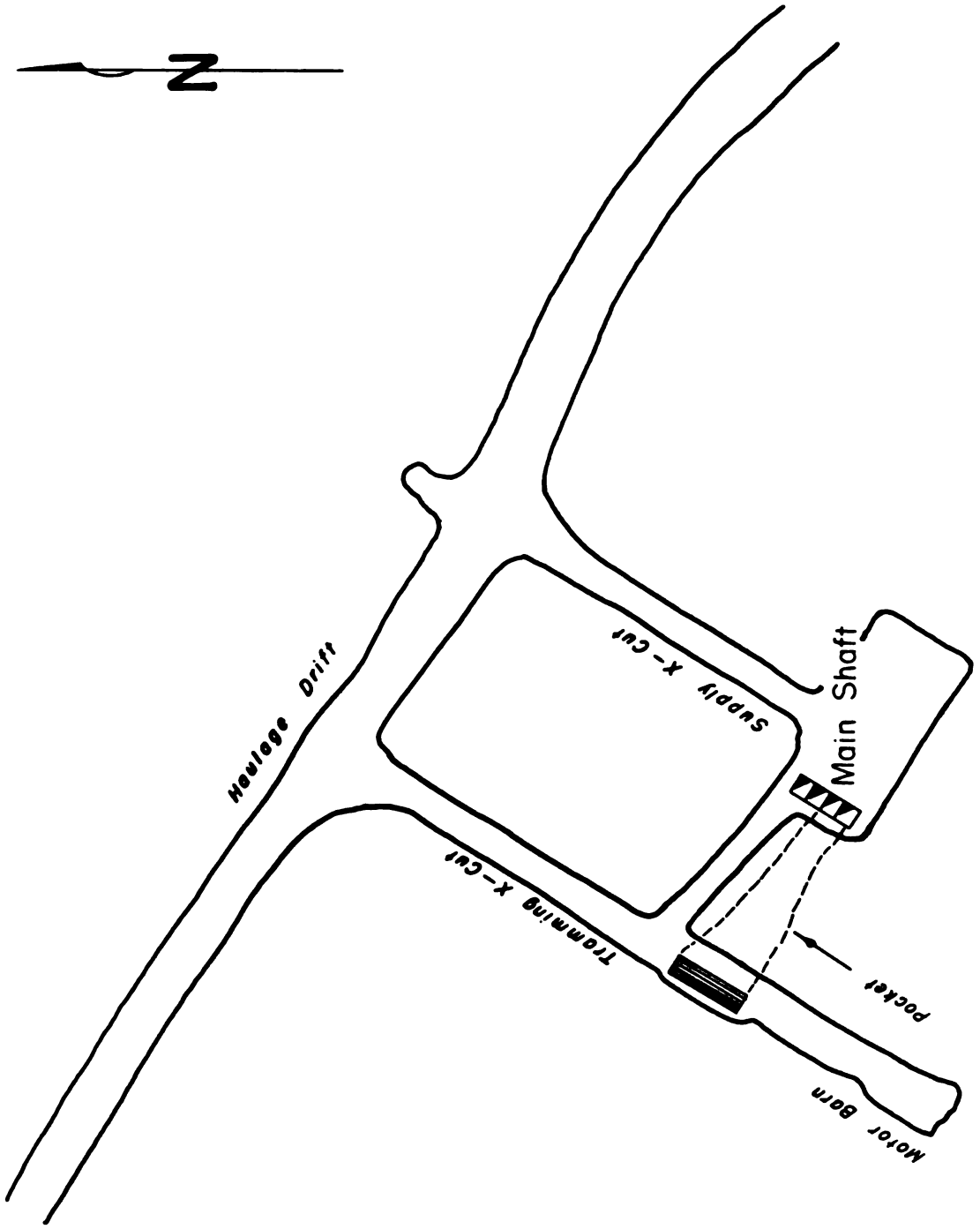

Figure 4. - Generalized plan of main shaft station.

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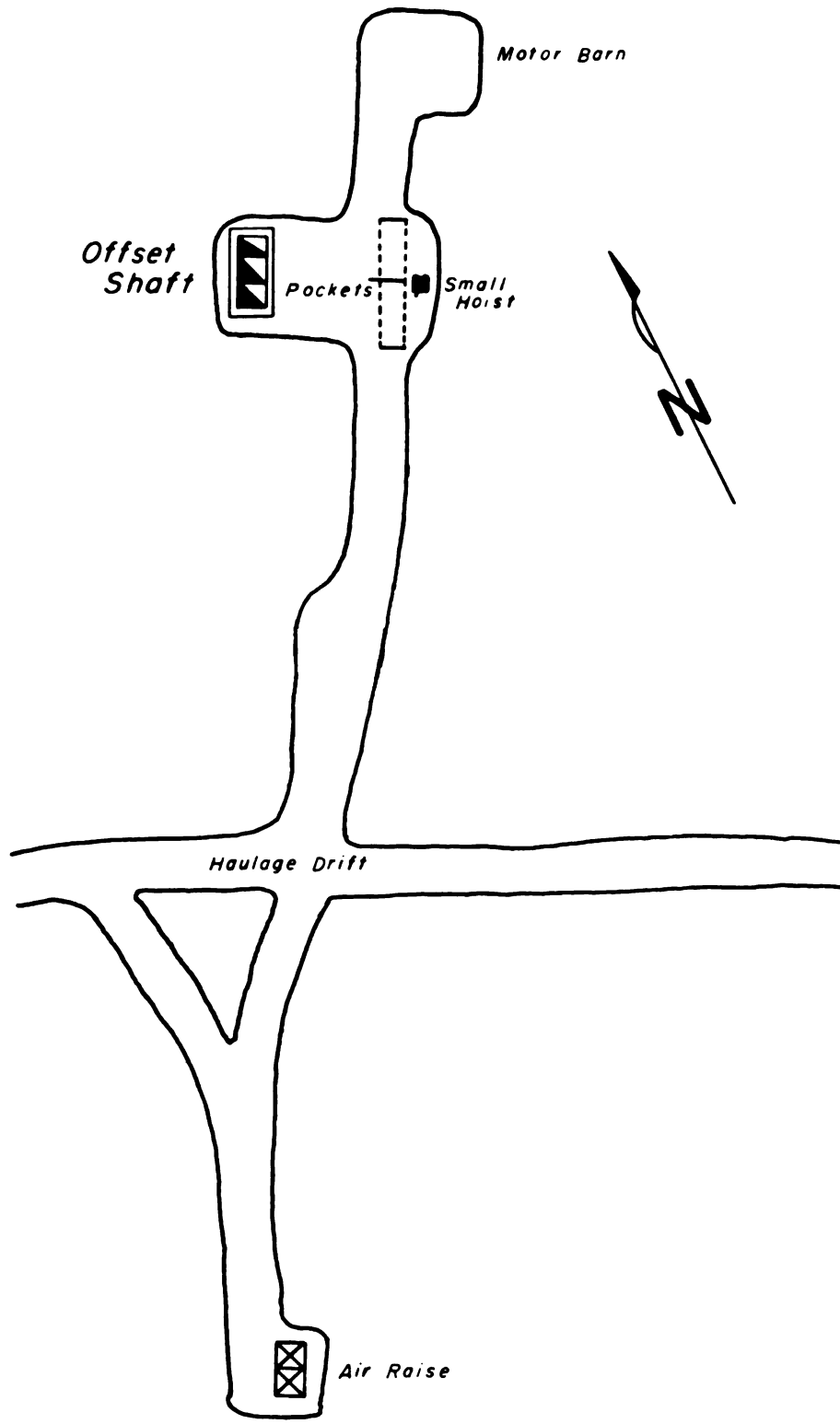

Figure 5. - Generalized plan of offset shaft station.

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The You Like vein, about 1,000 feet in the footwall of the Morning vein, was cut by No. 6 tunnel, the main haulage tunnel. It was developed further by 4 adits and several intermediate levels above the No. 6 tunnel and below by a vertical shaft to the 1,000-foot level. Below the bottom of this shaft the You Like was explored by crosscuts from the Morning mine on the 1,250-, 2,050-, 2,850-, 3,250-, and 3,650-foot levels.

Shaft Sinking

Main Shaft

Detailed data pertinent to the sinking of the main shaft are given in a Bureau of Mines report[6/] and information given here is summarized from that report.

The shaft consists of two main hoisting compartments, a supply compartment, and a pipe compartment. The size, measured outside timbers, is 23 feet 6 inches by 7 feet 3 inches. Standard 12-by 12-inch framed shaft timbers and 10-by 14-inch dividers were used with sets 5 feet center to center.

Shaft sinking was a continuous operation, with 4 men comprising a crew on each 8-hour shift. The entire bottom of the shaft was drilled and blasted, the usual round consisting of 48 holes 6 feet deep. Figure 6 shows the drill pattern for a main shaft round. The holes were drilled with light drifters on jack-hammer mountings using 1-1/4-inch conventional steel. The round was loaded with about 175 pounds of 40-percent gelatin dynamite and fired by electric delay blasting caps.

After a round was broken the men stood on the muck pile to timber, installing an average of one set per shift.

The broken rock was then hand-mucked directly into the sinking bucket until the muck pile at one end of the shaft was considerably lower than the rest of the pile. The remainder of the broken rock was mucked into a pan, which was dumped by a small hoist into the main sinking bucket for hoisting to the waste pocket on the level above. Each round of approximately 140 tons of rock required 4 to 5 shifts of mucking time.

Offset Shaft

When the limits of the main hoist were reached in 1937 it was decided to start a new offset shaft about 150 feet in the hanging wall on the 3,850-foot level. This vertical, 3-compartment shaft was sunk a lift at a time as production demanded and was bottomed at the sump below the 5,200-foot level.

The shaft was sunk through the pipe-manway compartment on a 3-shift continuous basis and did not interfere with normal production. A sinking crew consisted of 4 contract miners and 1 day's-pay hoistman per shift.

6/ See work cited in footnote 2.

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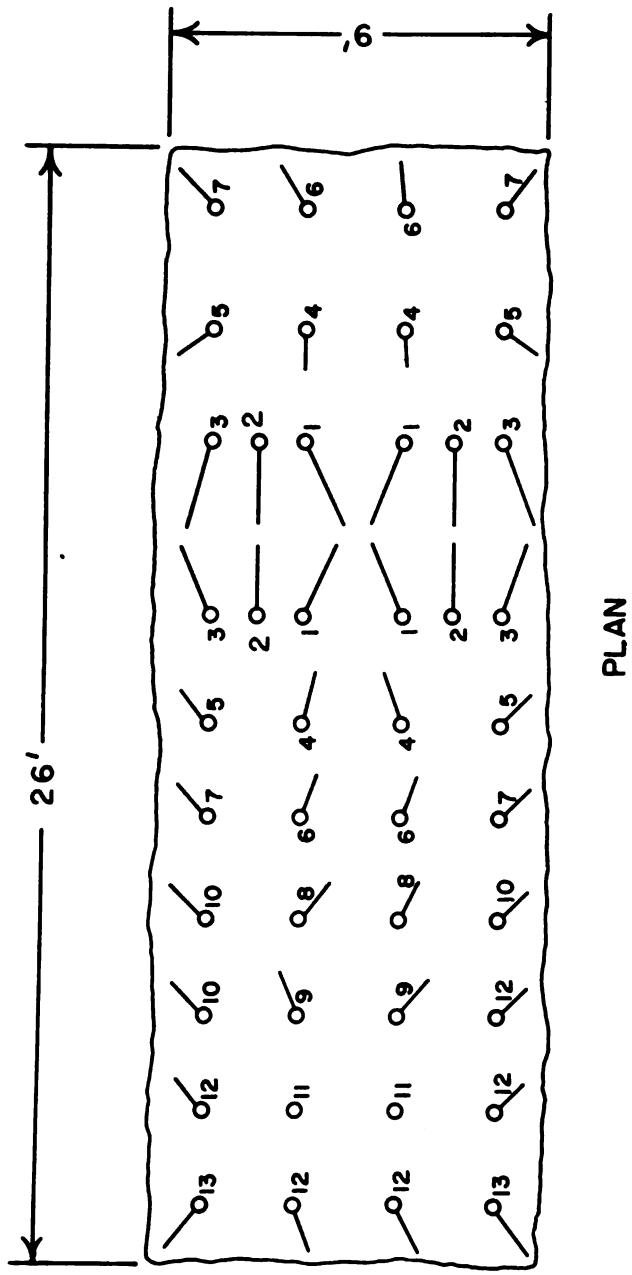

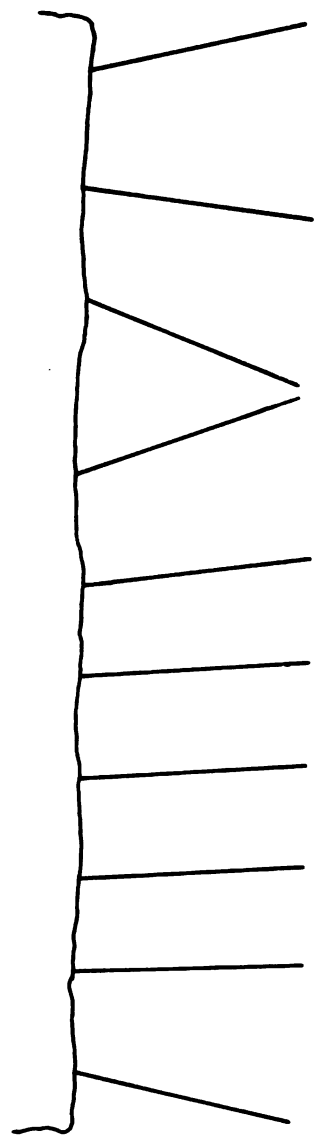

Figure 6. - Drill-hole pattern, main shaft round.

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The usual cycle of operations was as follows: Assuming a clean bottom, the crew drilled and blasted the half of the shaft nearest the pipe-manway compartment. When the first bench was mucked out the second bench was drilled and blasted. Timber was placed as soon as enough of the second round had been mucked to make room for it.

The last lifts of the offset shaft were sunk with 40-pound sinker-type rock drills, using integral steel with carbide-inert chisel bits. Starters were 2 feet 7 inches long, with 1-9/16-inch gage; seconds were 5 feet 3 inches long, with 1-3/8-inch gage; and thirds were 7 feet 10 inches long, with 1-1/4-inch gage. No set pattern was used for drilling the round, the miners using their own judgment as to the number and spacing of holes. An average bench round consisted of 26 to 30 holes 6 feet deep, as shown in figure 7. From 2-1/2 to 3 standard boxes of 40-percent gelatin dynamite detonated by standard electric delay blasting caps were used to blast each bench. Each complete shaft cycle broke an area approximately 19 by 9 feet.

Following blasting the round was shoveled directly into the sinking bucket until the corner under the bucket was mucked out. The remainder of the round was then shoveled or scraped into a pan powered by a small air hoist, which raised the pan when full and dumped it into the sinking bucket. The pan was refilled while the sinking bucket was making a round trip to the waste pocket on the level above. Mechanical mucking was considered impractical because timber had to be carried too near the bottom.

Timber headed with squeeze-type blocking and laced with 2-inch lagging (fig. 8) was placed as soon as enough of the second round had been mucked to make room for it. Both the main and offset shafts were sunk in heavy ground, and continuous relief was required to keep them open. Ground movement was such that within 2 years after sinking any section it usually was necessary to remove 2 to 3 feet of rock from all sides of the shaft and install jacket sets, as shown in figure 9. Continued squeezing made it necessary to replace jackets in 8 to 10 years.

Stations and Crosscuts

Main shaft stations are on the west side of the shaft down to and including the 2,250-foot level and on the east side of the shaft from the 2,250- to the 3,850-foot level. Crosscuts for servicing the levels were driven from the shaft stations north to the vein. To hold traffic congestion to a minimum, skip pockets were cut on the opposite side of the shaft from the stations and were also connected with the drifts by crosscuts.

Shaft stations and skip pockets for the offset shaft were on the east side of the shaft, and only one crosscut on each level was driven to the vein. Stations were generally cut the width of the shaft and 40 to 50 feet long, the height varying from 20 feet at the shaft to about 10 feet over the pocket. A small, air-driven hoist was used on each station to facilitate loading and unloading heavy timber and equipment.

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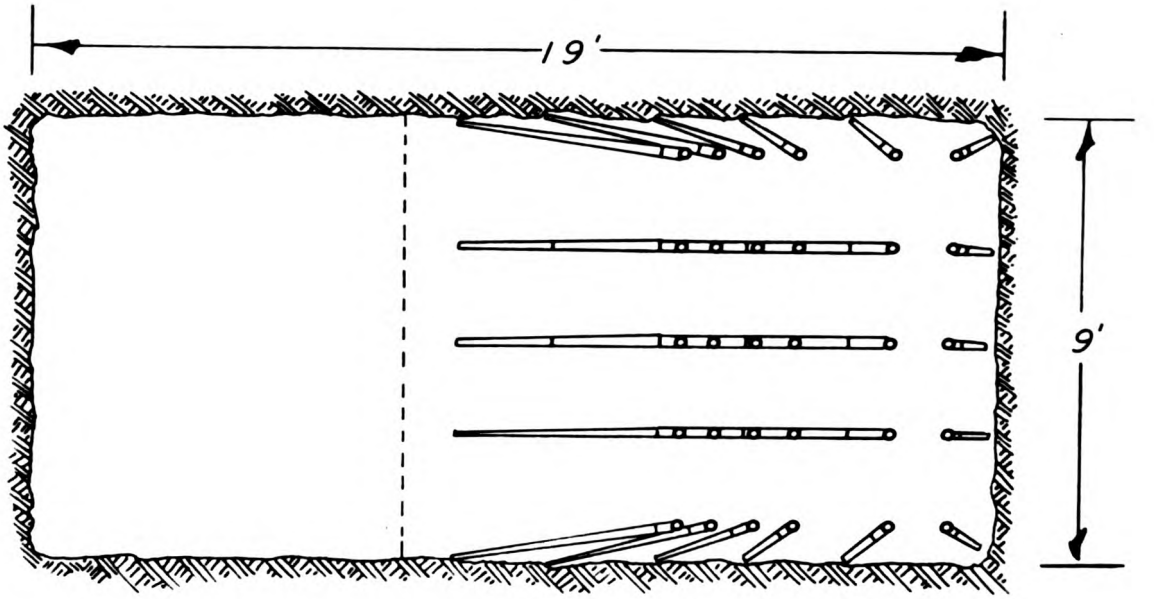
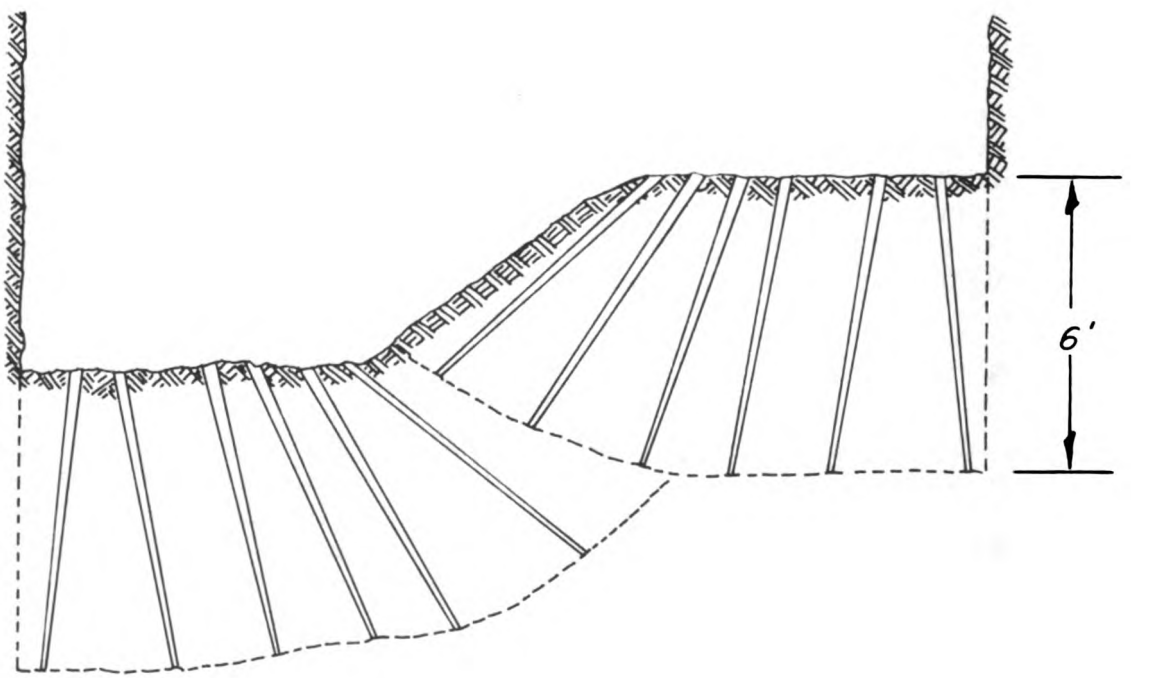


PLAN



VERTICAL SECTION SHOWING TWO SUCCESSIVE ROUNDS

Figure 7. - Drill-hole pattern, offset shaft.

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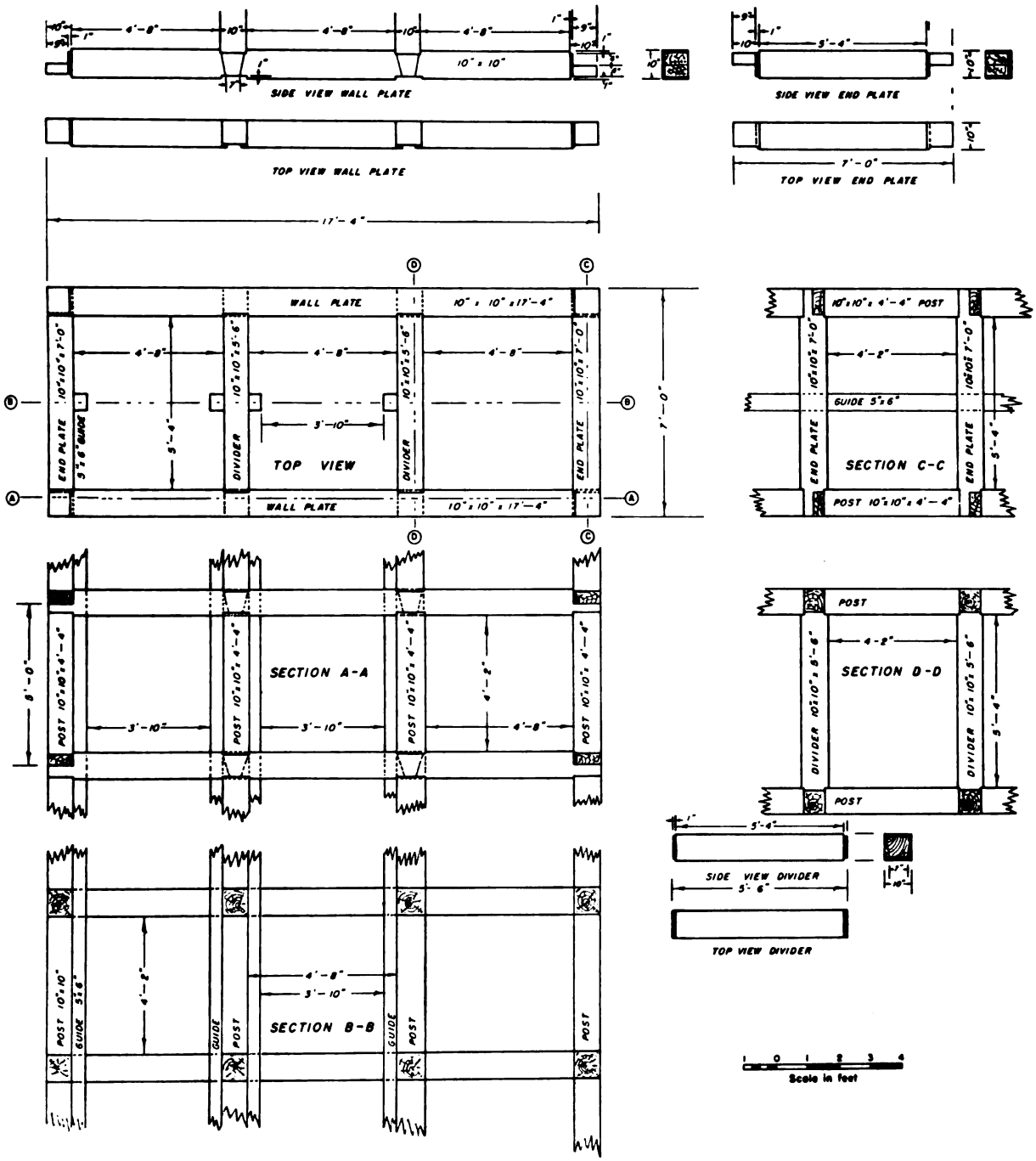

Figure 8. - Offset shaft timber.

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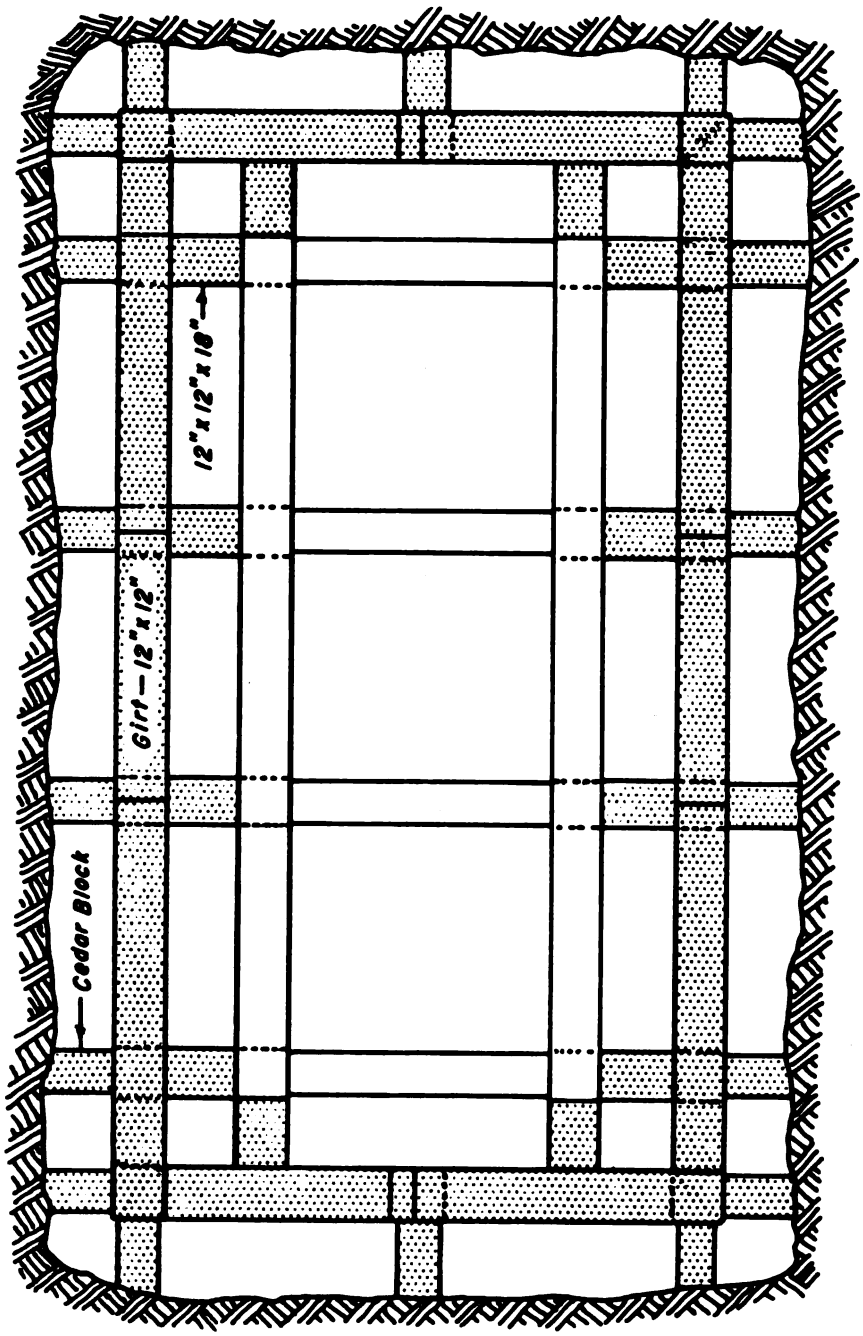

Figure 9. - Regular and jacket set, offset shaft.

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Crosscuts from the station to the vein were usually 9 by 9 feet in section and, wherever practical, were not timbered. Exploration crosscuts were 8 by 8 feet and seldom required timber.

Drifting

Development drifts were driven the full width of the ore and maintained as haulage drifts during the stoping cycle. Drift rounds were drilled with two 3-inch automatic drifters, mounted on a jumbo designed and built in the mine shop, using 1-1/8-inch round steel with detachable tungsten carbide bits. The drill pattern shown in figure 10 was varied somewhat in practice, the number of holes per round depending on the width of the ore. Drift rounds were blasted with 40-percent special gelatin dynamite.

Broken ore was loaded into 28-cu. ft. side-dump cars with an air-driven mucking machine from either a single or double track, depending on the width of the heading.

Miners were protected by stringers extending beyond the forward cap to within 1 foot of the face and covered with 3-inch lagging. The stringers were blocked; and, as the drift was advanced, the blocking was loosened and they were driven ahead; when extremely bad ground was encountered it was necessary to drive spiling.

All drifts were timbered with 3-piece, long-cap sets, bridging caps, and "basket" squeeze blocks or short stulls to the walls, as shown in figure 11. Precut on the surface to the extent possible, the main caps were 20-inch stulls slabbed on 3 sides and dapped on the bottom for the posts. Their size and weight made mechanical handling necessary, and they were lifted into place in the heading by the ingenious boom shown in figure 12, which was powered by the level haulage motor. When wall pressure started crushing the squeeze blocks, the swell was slabbed off and new blocks placed before the cap was broken. A small amount of timber recovered from the stopes was used for blocking, lagging, and cribbing.

Rock bolting was tried in place of the standard drift sets along a portion of the 5,200-foot level drift; however, the mine was closed before conclusive results of this experiment could be obtained. The management believes that rock bolting in sections of the vein too narrow or too low grade to mine was superior to timbering because it allowed a smaller cross section and required less drift and raise maintenance. Drifting was done on a 2-shift basis, usually by contract labor. For a typical drift round, designed to pull 5 feet, the average time required for each part of the drifting cycle was:

	Hours
Drilling and blasting	5
Mucking	5
Timbering	3

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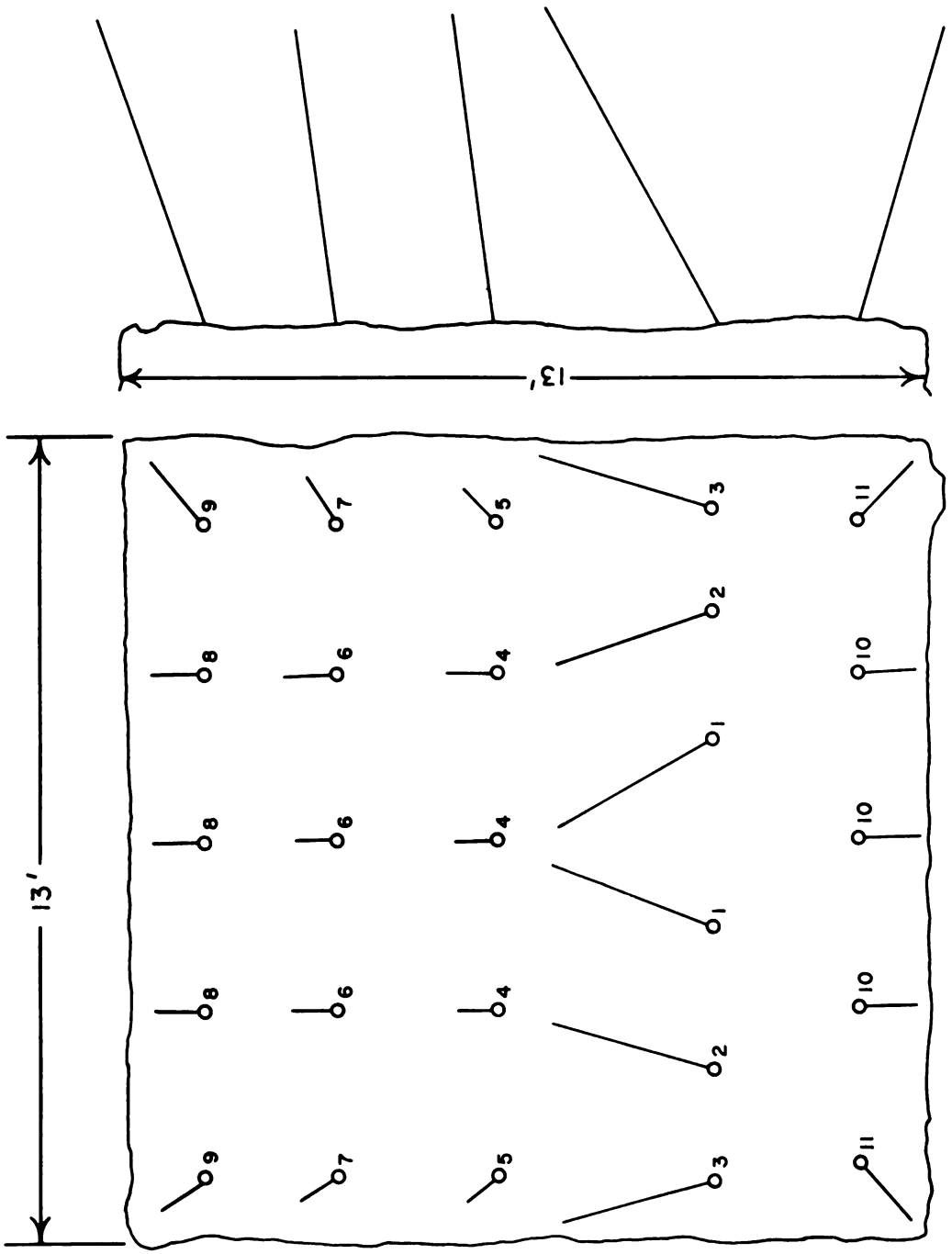

Figure 10. - Standard drift round.

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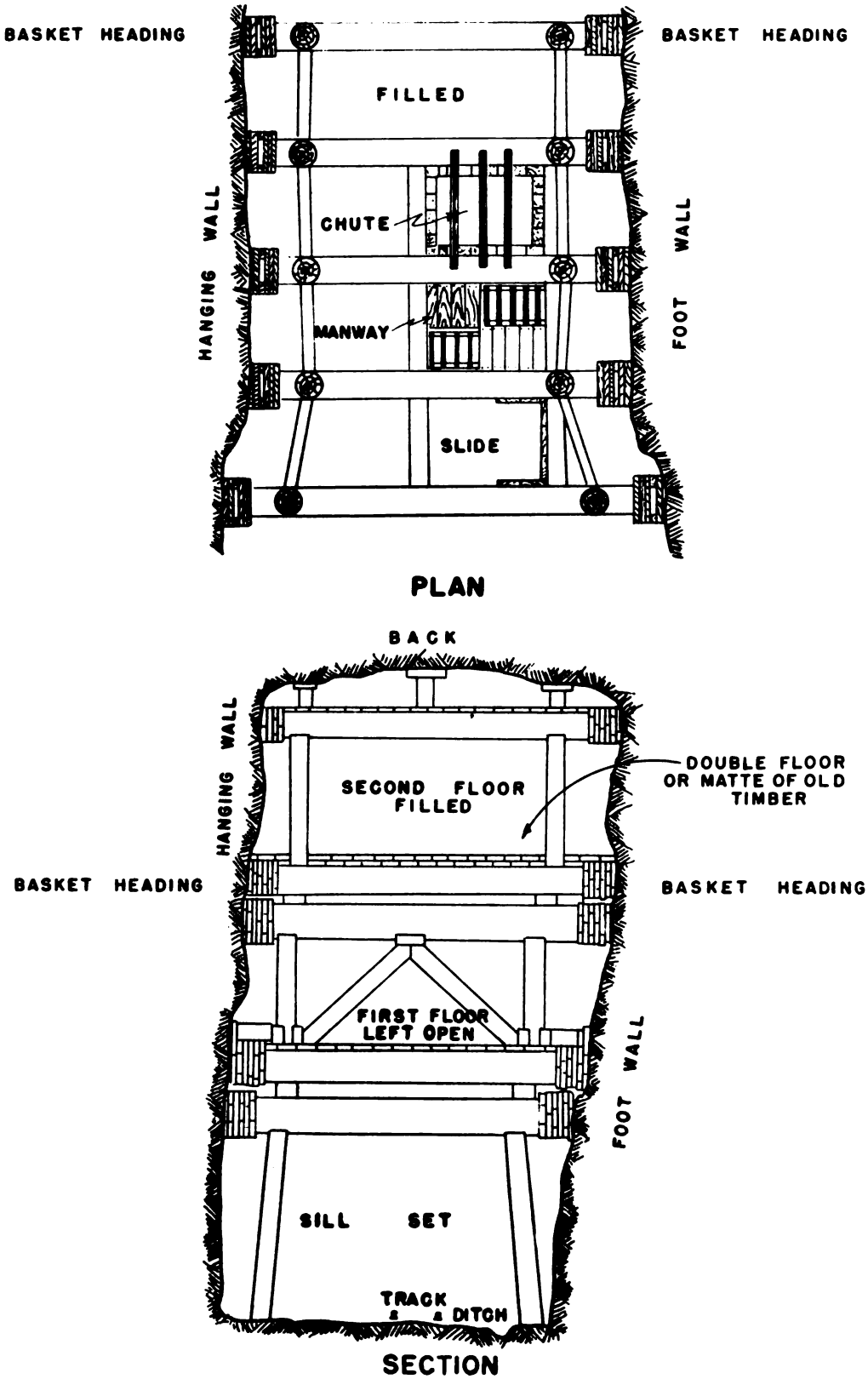

Figure 11. - Drift timber.

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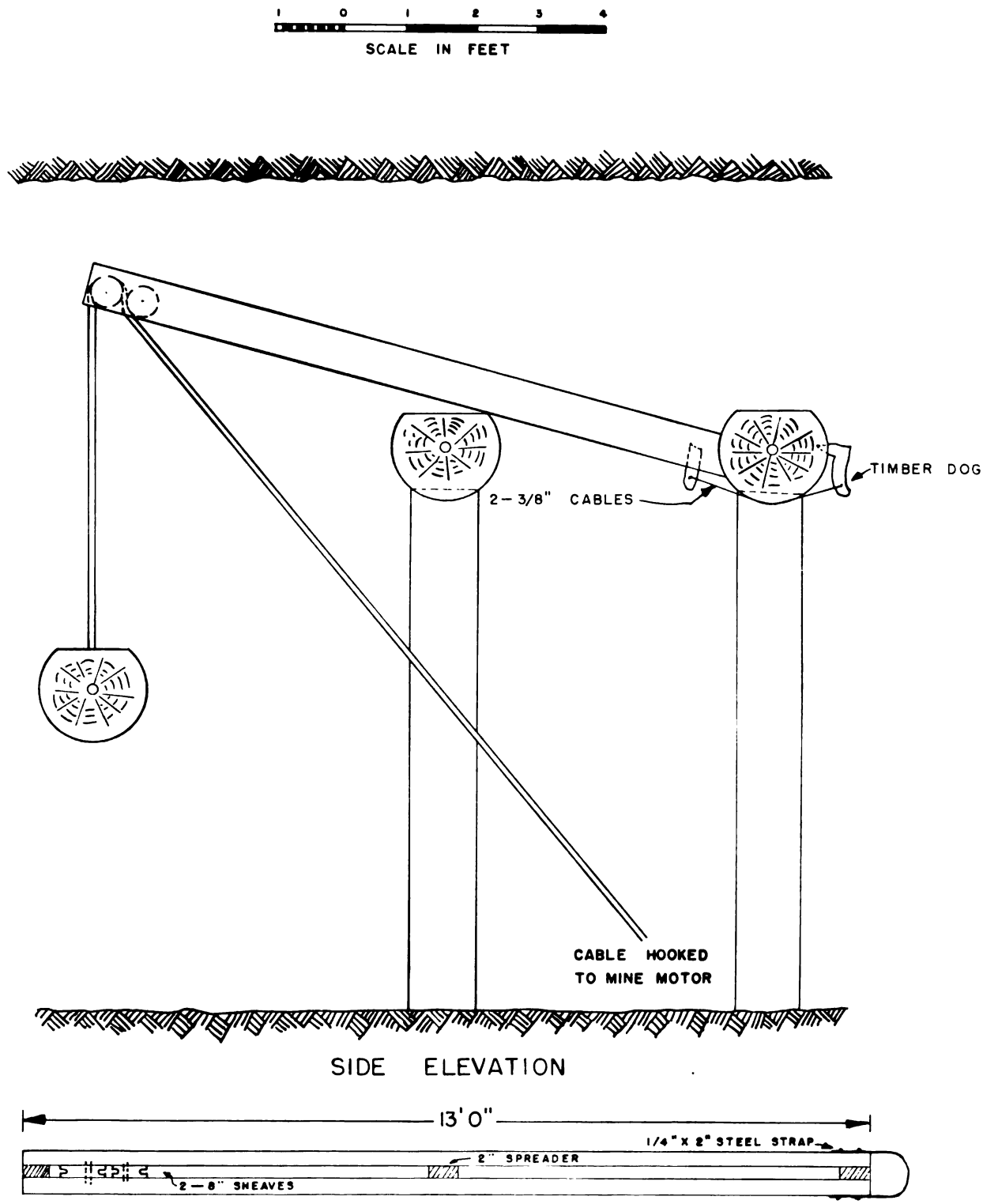

Figure 12. - Timbering boom, Morning mine.

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Raising

Four-cap, three-compartment development raises consisting of timber-slide, manway, and chute were carried the full width of the vein. Drilling was done with ordinary stopers using 1-inch quarter-octagon steel and detachable tungsten carbide bits. Because of high maintenance costs these raises were seldom carried more than one or two sets above the stope back and were advanced at about the same rate as the stope.

The timber sets were similar to those used in stoping and consisted of 4 caps 16 to 20 inches in diameter spaced 5 feet center to center with posts of round timber 10 inches in diameter and 8 feet in length. As soon as the raise had advanced 4 or 5 sets the posts were cut in the middle and intermediate caps put in. This operation was carried up at the same rate as stope filling. Maintenance of development raises was one of the most expensive items in mining.

Support requirements are well illustrated by a raise above the 4,650-foot level, which was not maintained during the war. To rehabilitate the stope it was necessary to install cribbing 7 feet thick on both sides of the raise, using logs 18 to 20 inches in diameter.

Stoping

Basically, stoping methods at the Morning have varied little throughout the history of the operation. Rock movement and heavy pressures made it necessary to support the walls immediately with timber after even a small cut. These wall conditions existed in the near-surface workings as well as in the lowest workings and to almost the same extent. The ore was easy to break, requiring few drill holes and a minimum of explosives.

Originally chutes were spaced at 25-foot intervals. Two floors of the stope, 1 for mucking and 1 for tramming, were left unfilled below the mining floor. Because the cost of maintaining chutes at this interval was excessive, an interval of 125 feet was adopted, and this practice was continued until the advent of mechanical slushing, when spacing at 65-foot intervals was found more satisfactory.

As shown in figure 13, stoping progressed by a series of horizontal cuts. Usually a stope was advanced one cut at a time; however, under ideal conditions two "staggered" cuts might be carried. The ore was drilled with automatic-feed drifters, using 1-1/8-inch round steel with detachable carbide bits and blasted onto lagging placed across the timber on the slushing floor. Broken ore was then slushed into the chute, drawn into 28-cu. ft. side-dump cars, and trammed to the skip pocket.

Before slushers were adopted the ore was blasted onto a mucking floor directly below the mining floor and hand-shoveled through movable steel slides into 1-ton cars on the tramming floor one set lower. The cars were hand-trammed to the chute and dumped.

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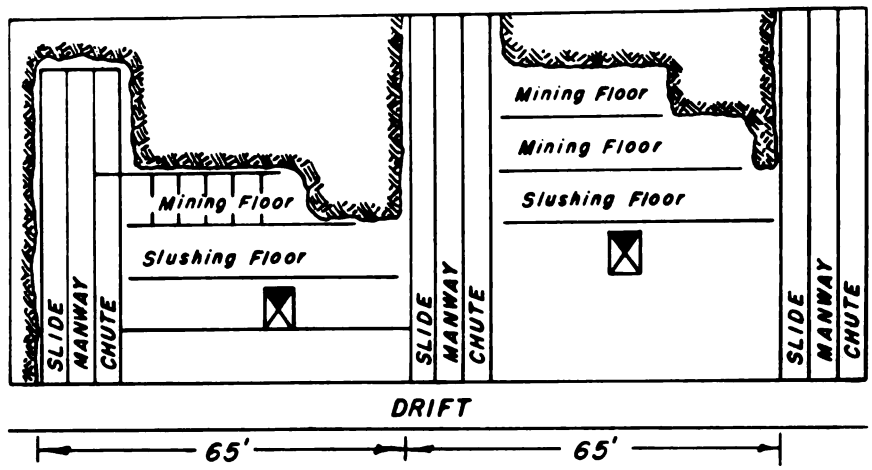

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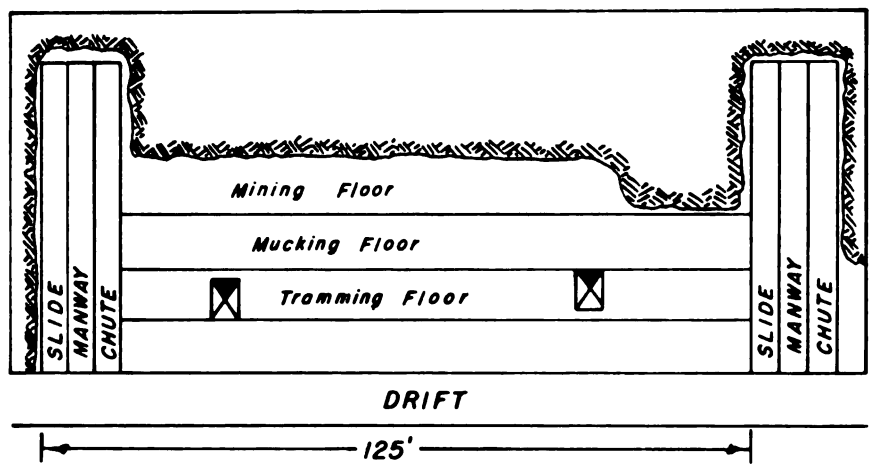

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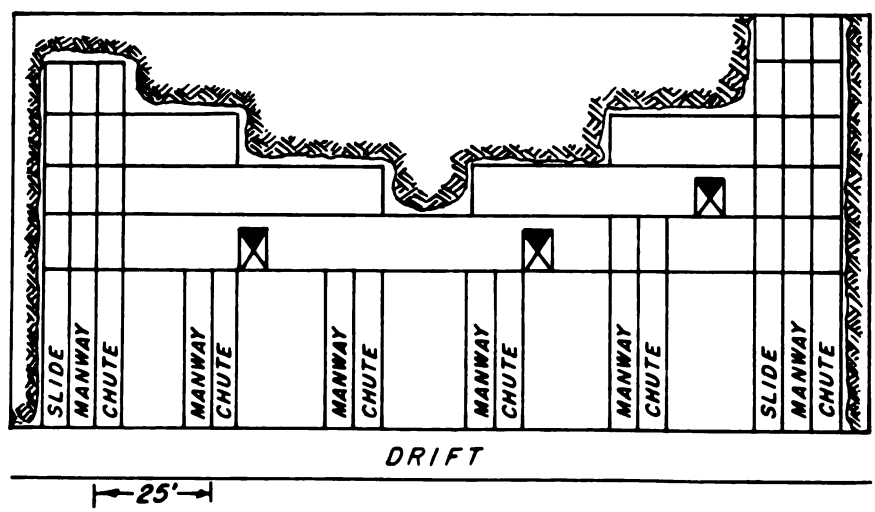

METHOD NO. 1.

Figure 13. - Stoping methods.

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Filling followed mining as closely as possible. As soon as a cut had been advanced to the next chute, the tramming or slushing floor was filled before a new cut was begun. Waste for filling was obtained from lateral drifts, waste crosscuts, ventilation raises, and waste raises in the walls, though in recent years the bulk came from horizontal crosscuts into the stope walls. These crosscuts began with a cross section of 5 by 7 feet, increased after 10 feet to 8 by 8 feet, and were driven 30 feet. When additional waste was needed the end of the crosscut was chambered horizontally. No timber was required except that necessary for reinforcing stope timber where the crosscut left the vein.

Stope sets were almost identical to those used in drifting. A single set consisted of 2 round posts, 10 inches in diameter and 8 feet in length, supporting a 16- to 24-inch-diameter cap as long as 18 feet. The cap length depended on the width of ore; and, where the stope was over 20 feet wide, butt caps were set in addition to regular caps. All caps were headed with basket squeeze blocks and were 5 feet center-to-center. Three-inch standard or slab lagging was used for lacing, as shown in figure 11.

Mining progressed as described above until the stope had been carried up to the 17th floor, where the unstable character of the remaining ore made it dangerous to proceed with standard methods. From this point the raise was completed to the level above, and the last 5 or 6 floors of the stope were mined by a procedure locally known as pulling sills.

This operation, illustrated in figure 14, was designed to provide maximum protection for the workmen. The timbering method was such that the miners always worked under timber and never under unmined ore. The numbers opposite the caps in the illustration show the sequence in which they were installed; many of these caps are in the filled area.

The ore was allowed to run into the chute by gravity until stoping had advanced to a point where mechanical handling became necessary. Then a slusher installed in the raise was used to remove ore from the stope. All timbering and servicing were done from the level above. Immediate filling was necessary, so as soon as the stope had been advanced 5 sets in a horizontal direction the 3 sets nearest the raise were laced off and filled. Only the slushing floor was left open.

Because of the heavy character of the ground, stopes were sometimes "lost" before the block could be completely mined. In general, a stope was considered lost when either of two conditions developed: (1) if extreme wall pressures made it impossible to keep the access raises open; and (2), if the ore began to cave, making it dangerous and costly to proceed in the conventional manner.

In recovering the remaining ore when a stope was lost, the usual method of attack was to drive a raise in either the footwall or hanging wall from one level to the next. Crosscuts were driven from the raise to tap the vein near the bottom and top of the block of ore being recovered. A raise was driven in the vein to connect the two crosscuts. The ore was then extracted by pulling sills. Timber

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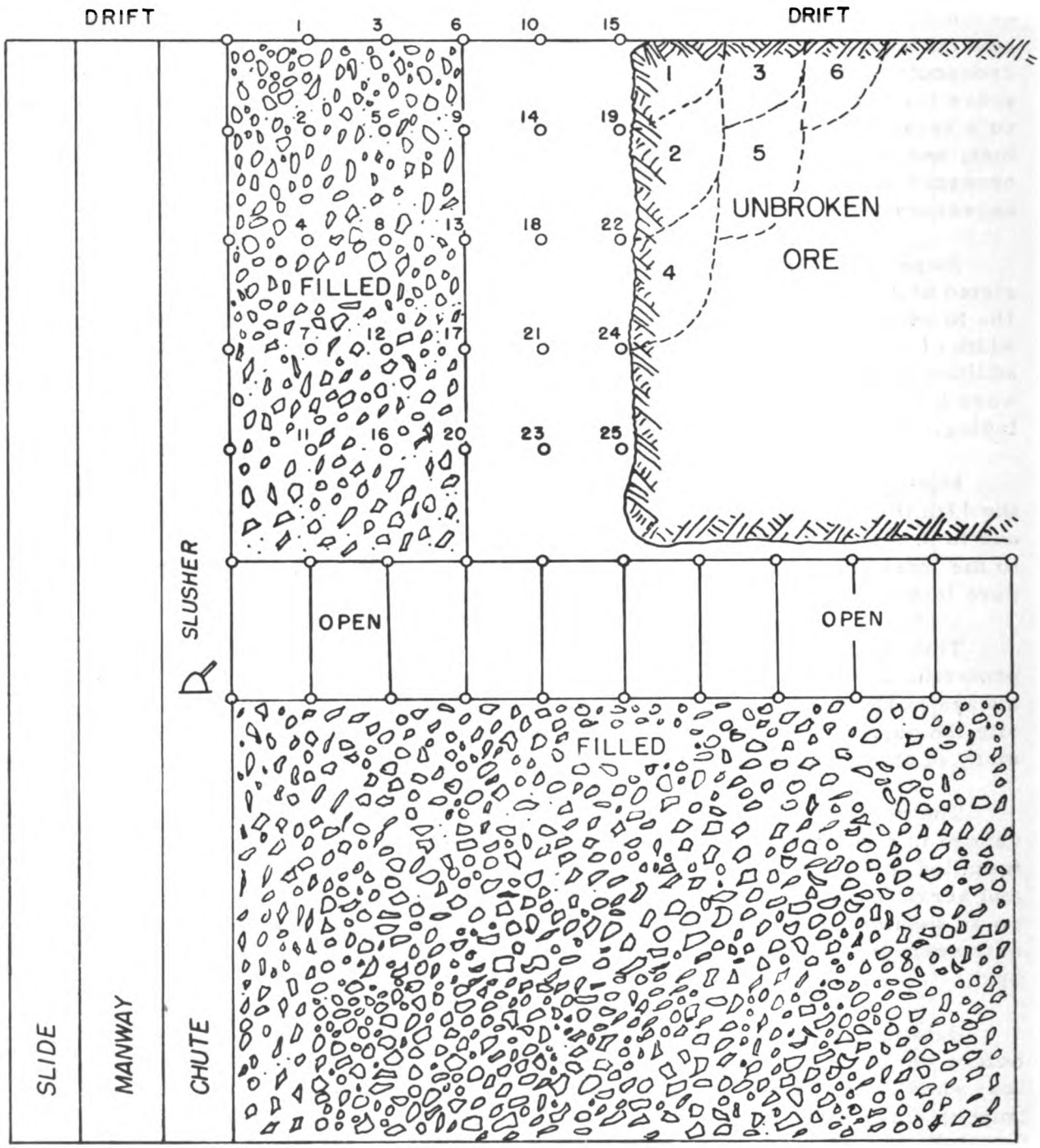

Figure 14. - Pulling sills.

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and supplies were taken into the stope through the upper crosscut, and the broken ore was slushed out through the lower. When the block was large, additional crosscuts were driven from the wall raise so that not too much ground would be open at one time. This method of recovery is illustrated in figure 15.

When the ore in a lost block caved by itself, a modified caving system was usually adopted. The block might be attacked from almost any direction. Access was gained through one of the mining floors, through a raise in one of the walls, or sometimes through a crosscut driven into the block from a footwall stope. Once access to both the bottom and top of the block had been provided, the ore was allowed to run. The caved ore was slushed from the bottom through whichever transfer working was available. Timber was installed from the top of the block downward at the same rate as the ore settled. This second method is shown in figure 16.

Transportation

Tramming on the levels was with 4-1/2-ton storage-battery locomotives, ten 28-cu. ft. side-dump cars making up a train. The cars were loaded from the stope chutes and trammed directly to the skip pockets, which were divided into ore and waste compartments. Skips with a capacity of 3-1/2 tons in the offset shaft and 4 tons in the main shaft were loaded from the pockets.

A double-reel hoist equipped with 1/2- by 6-inch flat cable and powered by a 600-hp. electric motor was used in the main shaft. Normally operated in balance at a rope speed of 1,300 feet per minute, this hoist had a capacity of 1,000 tons per 24 hours from the 3,850-foot level. A "chippy" hoist in the third compartment operated the cage for men, timber, and supplies.

The 8-foot-diameter double-drum hoist for the offset shaft had a capacity of about 800 tons in 24 hours. This 400-hp. motor-powered hoist was equipped with a 1-1/4-inch-diameter wire rope operated at a speed of 1,200 feet per minute. On the main level ore was trammed to the 1,200-ton mill bins in trains consisting of sixteen 60-cu. ft. hopper-bottom cars; waste was trammed to the waste dump between the river and the base of the mountain in trains made up of twenty to twenty-five 35-cu. ft. side-dump cars. All main-level trains were pulled by 6-1/2-ton, 550-volt, electric trolley locomotives.

Wherever possible material was handled mechanically. Lifttrucks were used extensively on the surface; flat timber was prepackaged at the main sawmill into bundles for convenient mechanical handling. Trolley locomotives similar to those underground were used to assemble supplies and materials from various shops, sawmills, and framing sheds. Underground the heavy timbers and supplies were loaded and unloaded with air hoists.

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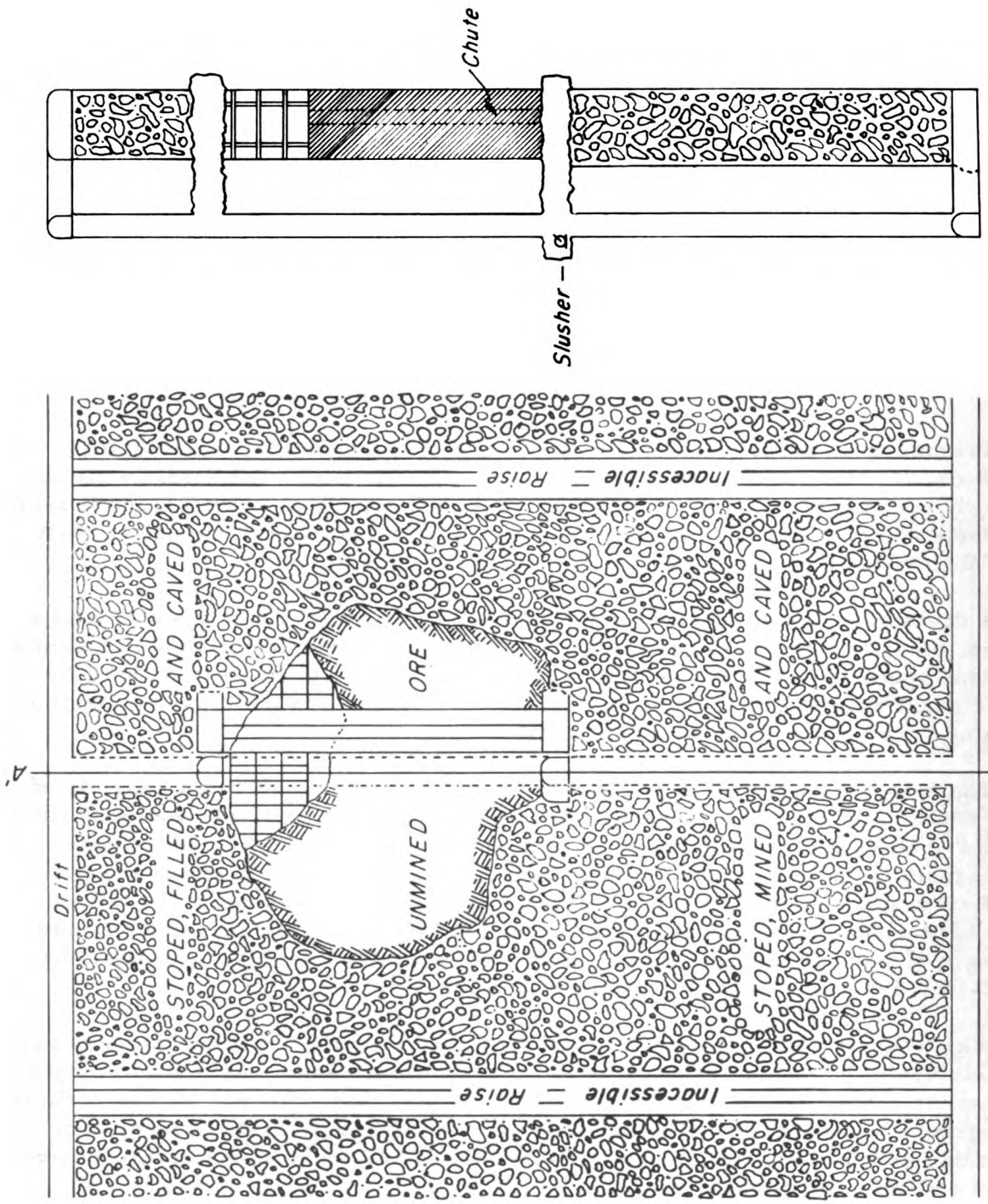

Figure 15. - Stope recovery method 1.

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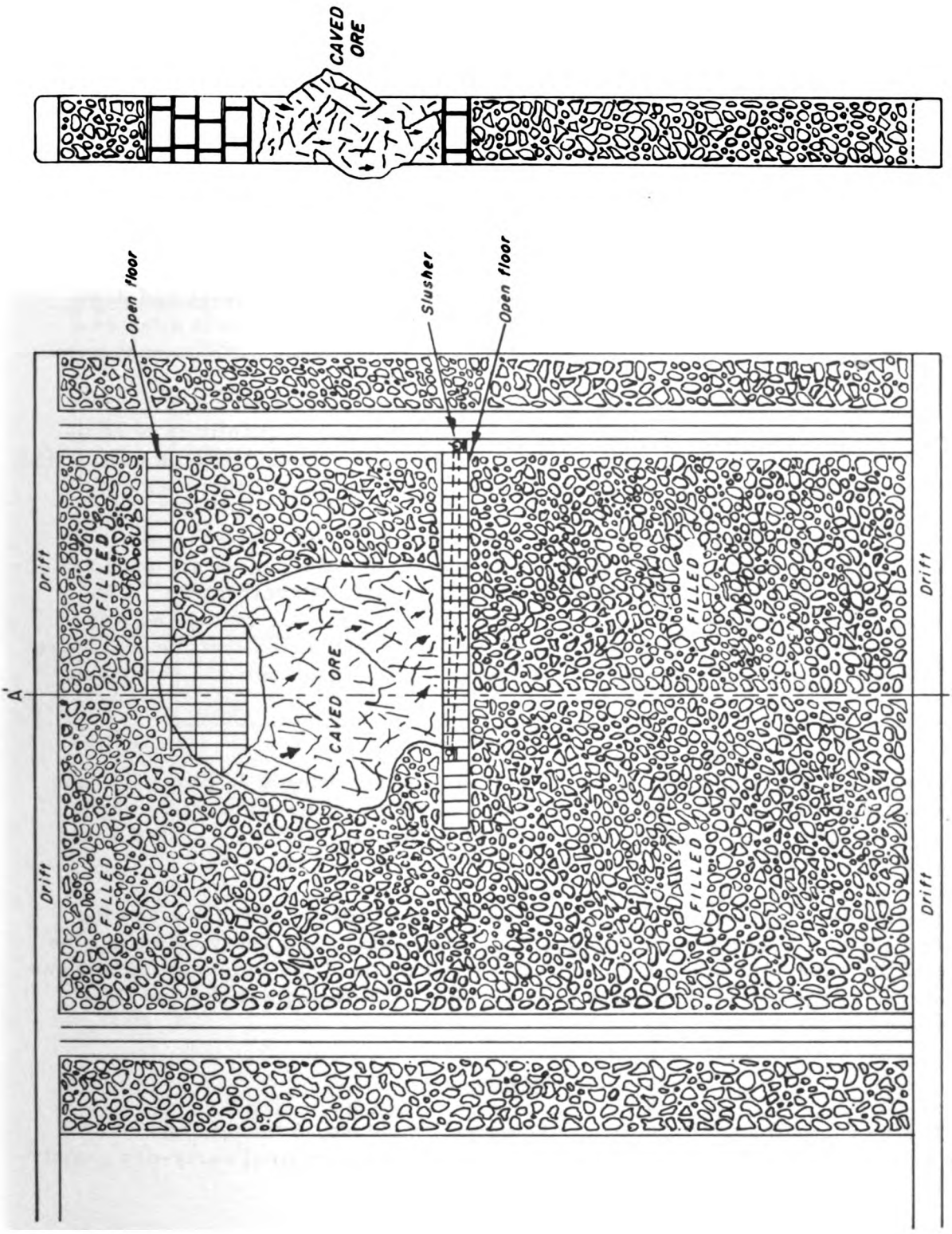

Figure 16. - Stope recovery method 2.

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Ventilation

Originally, two-compartment ventilation raises were driven in either the hanging wall or the footwall of the vein. They were generally untimbered except for a row of stulls to divide them into a manway-timber-slide compartment and a chute. To keep as far from the vein as possible, these raises were driven at an angle of about 45° away from the vein for half the distance to the next level and then turned and driven to connect with the level.

The ventilation system last used at the Morning is illustrated by figure 17. Fresh air entered the Morning workings from the Star mine on the Morning 3,050-foot level. It was then downcast through a series of ventilation raises to the 4,450-foot level, where a booster fan conveyed it along the level and down development raises to the 5,000-foot level. On the 5,000-foot level a fan exhausted the air up the shaft. From the collar of the shaft the air was exhausted through the No. 6 tunnel and through the upper workings.

Auxiliary ventilation was obtained by using unmeasured quantities of compressed air. It was used in the tops of stopes and other "dead" spaces to improve the air and reduce humidity. Below the 5,000-foot level there was no ventilation other than compressed air.

The temperature gradient was about three-fourths of a degree for each 100 feet of depth, and the rock temperature on the 5,200-foot level was 99°. The mine air was water-saturated; and, although the volume of air circulated was large, worker efficiency was low. The management estimates that the men were about 60 percent efficient compared with men working under normal conditions.

Organization

In addition to the Morning mine, the American Smelting & Refining Co. has several operating mines in the Coeur d'Alene district. The main office of the company northwestern mining department is at Wallace, Idaho, 7 miles west of Mullan.

The offices of the general management are at Wallace, Idaho, as are the employment, general engineering, geology, accounting, and assay offices. The office of the superintendent of concentrators and those of the mine superintendent and resident engineer, as well as the main shops, purchasing office, and time office, are at the mine.

Milling

The modern Morning mill, near the portal of the No. 6 tunnel, has a rated daily capacity of 1,200 tons. The structure housed the original early-day gravity-type milling equipment.

The first Morning ore was milled entirely by gravity, using jigs and tables. Later, Macquisten skin-flotation machines, installed to improve the zinc recovery, produced a concentrate assaying 45 percent zinc. This installation of

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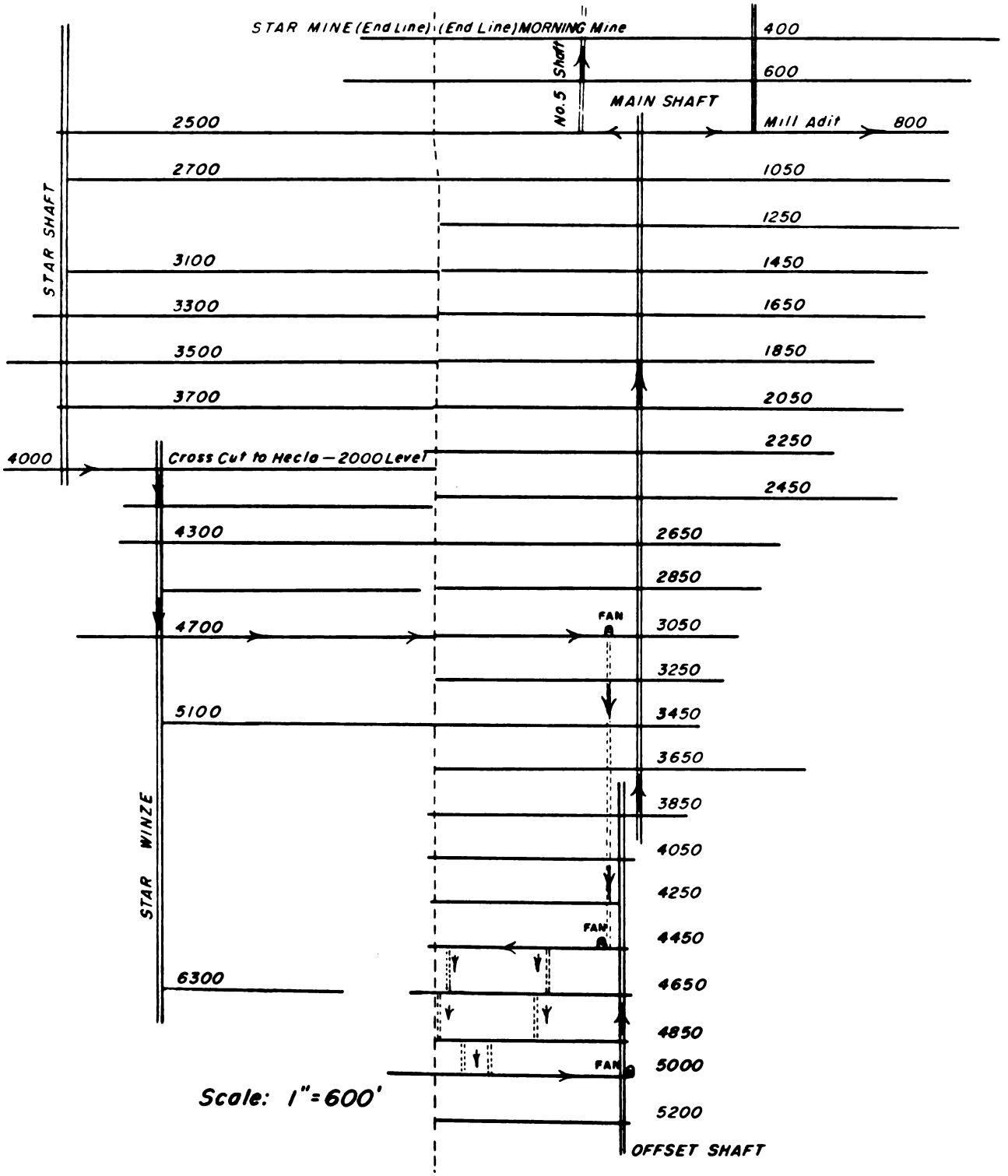

Figure 17. - Generalized plan of ventilation system, Morning and Star mines.

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Macquisten flotation machine was one of the world's largest and reportedly one of the most efficient.

The last of the jigs and tables were removed from the milling circuit in 1924, and by 1925 Owen & Dalton flotation machines were being used in the lead circuit and Cascade flotation machines in the zinc circuit. Although the mill had been completely converted from gravity to selective flotation, the sorting plant continued to operate in 1925, when it produced 14,794 tons of shipping ore containing an average of 11.9 ounces of silver, 28.7 percent lead, and 8.1 percent zinc per ton. The mill employed 91 men, operated 358 days during that year, and treated 374,399 tons of ore, an average of 1,045 tons per day. Metallurgical data for 1925, given in table 2, column 1, show results obtained during the first year the Morning mill operated using selective flotation exclusively.

TABLE 2. - Metallurgical data, Morning mill, 1925, 1939, and 1951

	1925			1939			1951		
Assay	Ag, oz./ton	Pb, percent	Zn, percent	Ag, oz./ton	Pb, percent	Zn, percent	Ag, oz./ton	Pb, percent	Zn, percent
Heads.....	4.1	9.0	5.3	2.9	7.6	5.7	1.79	5.72	8.32
Lead concentrate	24.9	58.4	11.4	26.7	77.6	4.0	19.7	74.7	6.4
Zinc concentrate	7.5	7.1	42.5	3.9	2.6	54.2	2.4	1.7	55.5
Recovery, percent	Ag	Pb	Zn	Ag	Pb	Zn	Ag	Pb	Zn
Lead concentrate	79.2	83.9	27.5	81.4	91.5	6.3	78.2	93.1	5.5
Zinc concentrate	11.0	4.7	47.4	12.4	3.1	88.9	18.4	4.0	92.1
Total	90.2	88.6	74.9	93.8	94.6	95.2	96.6	97.1	97.6

During 1939 the mill employed 57 men, worked 307 days, and processed 363,122 tons of ore, a daily average of 1,193 tons. Metallurgical data for 1939, the first year used for comparative purposes in this report, are given in table 2, column 2. The flowsheet was similar to that shown in figure 18.

The main difference between the milling practices in 1939 and 1951 was in the degree of grind. During the last several years the total mill feed has been ground to 80 percent minus-200-mesh and the sulfide portion further reduced to 50 percent minus-2,100-mesh, or 7 microns. A study of the metallurgical results for 1951 shows that this extremely fine grinding has not resulted in abnormal losses of lead, as is generally assumed to be the case where sliming is present, but rather in a remarkable degree of concentration and recovery of both lead and zinc.

By 1951 mine production was down, and the mill was run at only a fraction of its capacity. A crew of 29 men milled 88,817 tons of ore, an average of 319 tons per shift. Metallurgical data for that year are given in table 2, column 3.

Water for milling is pumped from a reservoir behind a dam on the South Fork of the Coeur d'Alene River. Requirements vary from a low of 150-200 g.p.m. to a maximum of 600 g. p. m.

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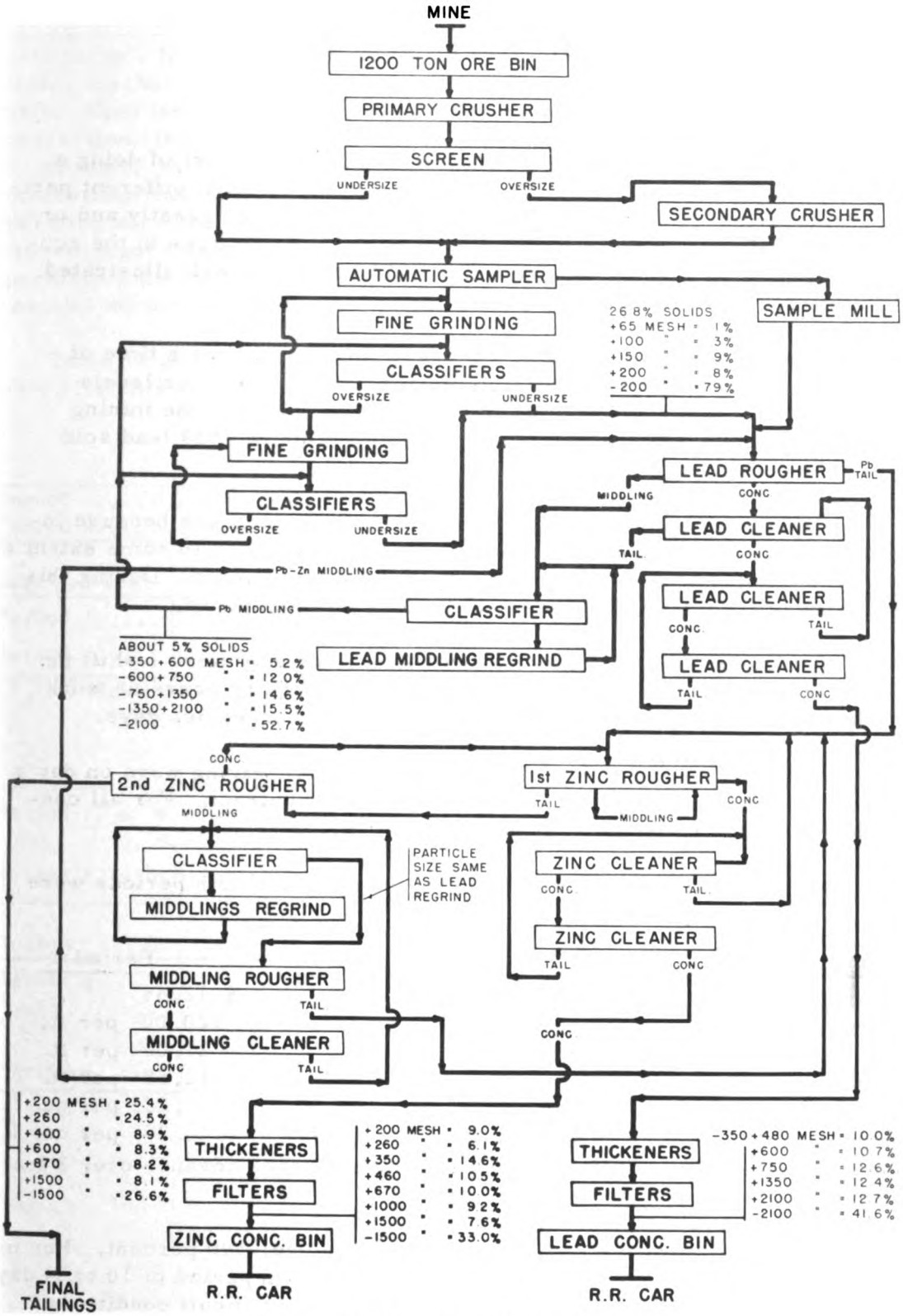

Figure 18. - Morning-mill flowsheet.

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Lead concentrates are shipped to the East Helena, Mont., smelter of the American Smelting & Refining Co. and zinc concentrates to the Anaconda Copper Mining Co. electrolytic zinc plant at Great Falls, Mont.

COSTS

The cost figures in the following tables show not only the cost of doing a specific job but also compare the costs for doing similar work at different periods in the history of the mine when economic conditions varied greatly and ore was being obtained from different depths in the mine. The changes in the economics of the Morning operation during the past 15 years are well illustrated by the cost data for the 2 periods selected.

The year 1939 was chosen for the first period because it was a time of transition when mining operations were being transferred to deeper levels being opened from the offset shaft. Metal prices were low, and the mining industry was still in a state of economic depression. During 1939 lead sold for 4.75 to 5.50 cents per pound.

Period 2, from August 1, 1950, to July 31, 1951, was chosen because increasing costs, increasing inaccessibility of ore reserves, and to some extent a decreasing rate of mining were making the operation uneconomic. During this period the price of lead ranged between 12 and 17 cents per pound.

In 1939 the mine worked 307 days on the basis of a 6-day week 1 shift per day 8 hours portal to portal. There was no change in working hours or work week between 1939 and 1950-51, during which the mine worked 302 days.

Actual stoping operations, tramming, hoisting, and repairing were on day's pay, but most of the development work was done under contract. For all contractors, however, the company guaranteed day's pay.

Day's pay wages and representative contract prices for both periods were as follows:

Item	Period 1	Period 2
Day's pay (miner).....	$6.00	$ 12.45
Shaft sinking.....	7.25 per ft. plus bonus 1/	120.00 per ft.
Drifting.....	.19 per cu. ft.	27.00 per ft.
Crosscutting	5.25 per ft.	12.50 per ft.
Stations.....	.15 per cu. ft.	.225 per cu. ft.
Skip pockets.....	.10 per cu. ft.	.25 per cu. ft.

1/ A bonus of 50 cents per shift was paid for each 3 inches advance over 2 feet per shift.

The average labor turnover during late years was over 100 percent. For men who had not worked before in the Morning mine, a break-in period of 10 to 14 days was required to permit them to become acclimated to the difficult conditions. Twenty-five percent of the new employees left before completing this break-in period.

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Tables 3 to 7, inclusive, show the direct cost of the different categories of development and tables 8 and 9 the total direct cost of mining and milling. These costs include labor and supplies, electric power, heating plant, compressor plants, mechanical shops, sawmills, trucks, tractors, and miscellaneous surface costs. Cost items that were not distributed and do not appear in these costs are supervision, laboratory expense, insurance, taxes, office salaries, engineering, and geology. As a rule, equipment costs were not distributed. Replacement rock drills, mucking machines, trammers, etc., were usually charged to the operating account. The entire cost of these items was written off at the time they were put into service. All equipment maintenance was charged to the operating account. Items upon which depreciation was figured were carried in a capital account, which was not distributed.

Table 10 is a balance sheet showing the entire cost of the Morning operation; table 11 shows the costs of a few representative supply items for each period.

TABLE 3. - Shaft-sinking costs

Period 1..... Offset shaft; 19- x 9-foot cross section, 208-foot advance 1/
Period 2..... Offset shaft; 19- x 9-foot cross section, 85.5-foot advance

Cost per foot	Labor	Timber	Explosives	Other supplies	Power	Total
Period 1.....	$ 61.90	$ 7.30	$ 7.48	$11.79	$2.22	$90.68
Period 2.....	163.42	30.92	11.51	21.52	2.82	230.19

1/ No sinking during 1939; figures are for 1938.

TABLE 4. - Station and pocket costs

Period 1. a. 4,250-foot-level station; total excavation, 13,990 cu. ft.
b. 4,250-foot-level motor barn and sump; total excavation, 4,630 cu. ft.
Period 2. 5,000-foot-level station and skip pocket; total excavation, 20,750 cu. ft.

Cost per cu. ft.	Labor	Timber	Explosives	Other supplies	Power	Total
Period 1:						
a.	$0.168	$0.015	$0.012	$0.025	$0.002	$0.222
b.	.264	-	.024	.023	.006	.318
Period 2	.509	.063	.053	.086	.015	.726

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TABLE 5. - Crosscut and lateral costs

Period 1. 4,250-foot level; 8- x 9-foot cross section, 239-foot advance, total excavation 17,274 cu. ft.

Period 2. 5,000-foot level; 9- x 9-foot cross section, 549-foot advance, total excavation 51,000 cu. ft.

	Labor	Timber	Explosives	Other supplies	Power	Total
Period 1:						
Cost/ft.	$ 8.84	$0.20	$2.07	$3.26	$0.58	$14.96
Cost/cu. ft.	.122	.003	.029	.045	.008	.207
Period 2:						
Cost/ft.	20.08	.13	3.87	7.23	1.22	32.53
Cost/cu. ft.	.216	.001	.042	.078	.013	.350

TABLE 6. - Drifting costs

Period 1. 4,050-foot level; 13- x 14-foot cross section, 615-foot advance, total excavation 115,479 cu. ft.

Period 2. 5,000-foot level, 11- x 13-foot cross section, 430-foot advance, total excavation 61,500 cu. ft.

	Labor	Timber	Explosives	Other supplies	Power	Total
Period 1:						
Cost/ft.	$14.97	$3.40	$2.97	$4.02	$0.74	$26.10
Cost/cu. ft.	.080	.018	.016	.021	.004	.139
Period 2:						
Cost/ft.	39.61	5.92	3.88	8.61	1.22	59.24
Cost/cu. ft.	.277	.041	.027	.060	.009	.414

TABLE 7. - Stoping costs 1/

Period 1. 249,400 tons mined by company

Period 2. 93,637 tons mined by company

Cost per ton	Labor	Timber	Explosives	Other supplies	Power	Total
Period 1....	$1.44	$0.30	$0.09	$0.14	$0.19	$2.16
Period 2....	6.12	.82	.17	.79	.57	8.48

1/ Mining conditions were more severe in 1951; this is partly responsible for increased stoping costs.

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TABLE 8. - Mining costs, company ore only

	Tons	Labor	Supplies and power	Total
Period 1 (1939).....	249,400	$871,693.01	$348,720.41	$1,220,413.42
Cost per ton.		3.495	1.398	4.893
Period 2 (1950-51)...	93,637	1,303,599.99	448,777.23	1,752,377.22
Cost per ton.		13.922	4.793	18.715

TABLE 9. - Total milling cost[1/]

Period	Source	ore milled	Labor	Supplies and power	Total
1	Company.....	249,400			
	Leasers.....	113,722			
	Total.....	363,122	$105,358.39	$203,732.31	$309,090.70
	Cost per ton		.290	.561	.851
2	Morning mine..	96,858			
	Other mines...	75,020			
	Total.....	171,878	147,365.13	199,480.52	346,845.65
	Cost per ton		.857	1.161	2.018
	Revenue from custom milling				150,040.00
	Net Morning cost				196,805.65
	Net cost per ton				2.032

1/ For additional cost data see: Dalton, M. P., Milling Methods and Costs at the Morning Concentrator of the Federal Mining & Smelting Co., Mullan, Idaho: Bureau of Mines Inf. Circ. 6587, 1932, 11 pp.

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TABLE 10. - Total cost distribution

Mining	Period 1	Period 2
Ore mined, tons:		
Company	249,400	93,637
Leasers	113,722	3,221
Total	363,122	96,858
Based on ore mined by company only:		
Superintendency plus vacations except develop. and shaft maint.	$42,118.70	$53,464.43
Cost per ton	.17	.57
Stoping	538,281.92	793,798.27
Cost per ton	2.16	8.48
Tramming on levels	60,203.03	88,312.86
Cost per ton	.24	.94
Hoisting	101,969.84	151,427.41
Cost per ton	.41	1.62
Haulage on main tunnel	24,790.47	55,152.60
Cost per ton	.10	.59
Drainage	15,059.26	49,677.90
Cost per ton	.06	.53
Shaft and station maintenance	144,679.91	228,766.20
Cost per ton	.58	2.44
Ventilation	23,559.61	31,343.88
Cost per ton	.29	.33
Lighting	8,734.43	7,940.03
Cost per ton	.03	.08
General underground maintenance	118,859.58	95,261.64
Cost per ton	.48	1.02
Miscellaneous	49,823.98	93,434.21
Cost per ton	.20	1.00
Development	80,577.14	79,638.43
Cost per ton	.32	.85
Mine dry and headhouse	11,755.55	24,159.36
Cost per ton	.05	.26
Paid to leasers	436,436.94	41,768.50
Total mining cost	$1,656,850.36	$1,794,145.72
Cost per ton	4.563	18.523
Milling		
Tons milled from Morning mine	363,122	96,858
Tons milled from other mines	-	75,020
Total tonnage milled	363,122	171,878

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TABLE 10. - Total cost distribution (Con.)

Milling (Con.)	Period 1	Period 2
Superintendency plus vacations	$12,705.11	$20,522.15
Cost per ton	.03	.12
Crushing	19,885.99	25,230.75
Cost per ton	.05	.15
Grinding	80,571.19	89,707.32
Cost per ton	.22	.52
Flotation	132,663.26	120,150.12
Cost per ton	.37	.70
Filtering	12,536.77	13,961.94
Cost per ton	.03	.08
Sampling	430.30	2,105.18
Cost per ton	-	.01
Heating and lighting	13,738.43	32,227.33
Cost per ton	.04	.19
Water supply	1,807.76	3,664.53
Cost per ton	-	.02
Building maintenance	9,599.00	6,069.53
Cost per ton	.03	.04
Ore loading	5,609.62	4,663.73
Cost per ton	.02	.03
Mill dry	1,994.46	790.03
Cost per ton	.01	.01
Carpenters and machine shop, not distributed	4,762.19	7,574.65
Cost per ton	.01	.04
Miscellaneous	12,786.62	20,178.39
Cost per ton	.04	.12
Power	Distributed	Distributed
Cost per ton		
Total milling cost	$309,090.70	$346,845.65
Cost per ton	.851	2.018
Total Morning milling cost	309,090.70	196,805.65
Cost per ton	.851	2.032
General expenses		
Tons ore mined	363,122	96,858
Laboratory	$ 16,103.65	$ 3,176.27
Cost per ton	.04	.03
Insurance	79,811.18	82,313.48
Cost per ton	.22	.85
Taxes, state, local, and federal	76,514.78	88,749.54
Cost per ton	.21	.92
Office salaries, including engineers	24,326.54	46,808.94
Cost per ton	.07	.48
Miscellaneous	98,863.30	100,664.70
Cost per ton	.27	1.04
Total general	$ 295,619.45	$ 321,712.93
Cost per ton	.814	3.321
Total, mining, milling, and general	$2,261,560.51	$2,312,664.30
Cost per ton	6.228	23.877

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TABLE 11. - Cost of supplies

Item	Unit	Period 1	Period 2
Powder	50-lb. box	$ 6.25	$ 9.68
Lagging	M	19.00	65.00
Stull	16-in. diam./ft.	.1872	.5635
Rail	2,240 lb.	55.45	176.62
2-in. pipe	100 ft.	16.09	39.04
1-1/8-in. drill steel	100 lb.	12.59	20.48
Grinding balls	100 lb.	3.73	5.75

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